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Ourselves.

Seven years ago this journal was founded as "Electrochemical Industry," to be the exponent of electrochemistry and its industrial applications. It has grown as these branches have developed, has put forth its best endeavors to represent every phase of this development, and will increase rather than relax in its future service to this great field of professional and industrial activity. Owing to the overlapping of, and the interrelations between, the different branches of chemistry and metallurgy, however, it was early necessary to enlarge the scope of the editorial policy so as to include more broadly the industrial applications of chemistry and metallurgy, and more particularly those of an engineering nature. This evolution first found expression in the change of name from "Electrochemical Industry" to "Electrochemical and Metallurgical Industry" in 1905, and a further step was the absorption of the "Iron and Steel Magazine" in 1906. The change of name now announced to "Metallurgical and Chemical Engineering" is the final consequence of the logical development of the past seven years, for the new title defines precisely the field which this journal has been more and more making its own in its past career. The adoption of the new name brings us privileges and lays upon us obligations—the privilege of addressing more generally than before all men interested in the engineering features of metallurgy and chemistry, and the obligation of living up to our responsibility to cover as completely as possible all important phases of these broad fields.

* * *

The work to be done is well defined and the field is not covered by any other journal. We do not address the purely mining engineer on the one hand nor the purely analyst on the other. In brief, all our efforts will be concentrated on making the best possible journal to serve the metallurgical and chemical sides of engineering. There is a natural community of interest between the metallurgical and the chemical engineer. Managers and engineers of ore concentration plants, of cyanide works, of metal smelters, of refineries, as well as manufacturing chemists and chemists engaged in engineering practice in industrial establishments of all kinds, are all more or less interested in the same processes—in crushing and grinding, concentration and separation, drying and evaporating, filtration, calcination, combustion, furnace operation, fusion, welding, in the measurement and control of temperatures, in chimney draft and forced draft, and in a hundred other lines of work. By making the word "Engineering" part of our new title, we wish to emphasize the fact that this journal is not, on the one hand, a trade paper, nor, on the other hand, a periodical representative of pure science. While neither commercial news nor scientific articles or abstracts will be absent from our columns, they will be secondary to the record and discussion of what is primarily needful in the work of the engineer or in the conduct and management of metallurgical and chemical works. Naturally, in these days when electrochemistry is extending more and

more in its scope and in its relations with other branches of metallurgy and chemistry—we need mention only the electric refining furnace in the steel industry and the fixation of atmospheric nitrogen in the manufacture of fertilizers and explosives—it will be our duty and our privilege to be as jealous an exponent of electrochemistry in all of its phases as ever. This is our birthright, which we do not relinquish by change of name. But along the broader lines above sketched, we feel convinced that this journal can be made a powerful catalytic agent in the advance of metallurgical and chemical engineering; and we sincerely hope that in our efforts to the attainment of this end, we may count on the support of all interested in the cause in which these pages are enlisted.

Again the Problem of Electrode Losses

The interesting article by Dr. J. Forssell of Sweden, published in this issue, had been written before the author had seen the article of Mr. Hering in our October issue and the papers on the same subject, presented at the last meeting of the American Electrochemical Society, and published in our December issue. This explains the omission of references to these publications in Dr. Forssell's article. While the treatment of the subject by Dr. Forssell has various original features, it is particularly interesting on account of the way in which he takes account of the heat leakage from the electrodes into the wall. He lumps this together with half the Joulean heat loss and introduces into his formulas a coefficient which may vary from ½ to 1 according to conditions. The method is feasible. But to get a simple conception of what really happens, it is perhaps preferable to consider the matter in the following way, the results being easily proven by analysis, at least under simplifying assumptions. We can consider the whole heat loss as a superposition of three phenomena. Firstly, we have the simple heat conduction, due to temperature difference at the two ends; this is the heat H that would flow if there was no electric current. Secondly, we have the Joulean heat i^2r. Thirdly, we have the heat leakage L from the electrode into the surrounding wall. The Joulean heat flows off impartially to both sides, ½ i^2r into the furnace and ½ i^2r into the water jacket outside. The heat L which is given off from the electrode to the surrounding walls is also derived impartially in equal halves from both sides, ½ L from the interior of the furnace and ½ L from the water jacket. The heat which flows from the furnace into the hot electrode end is, therefore, $H-\frac{1}{2}i^2r+\frac{1}{2}L$. The heat which goes into the water jacket is $H+\frac{1}{2}i^2r-\frac{1}{2}L$. But this latter quantity is not the whole electrode loss, since the latter includes that heat which is given off from the electrode to the furnace walls. If this goes completely to the outside, the total electrode loss is $H+\frac{1}{2}i^2r+\frac{1}{2}L$. Further advance must now be made experimentally. It would seem advisable to determine the "constants" for carbon and graphite under conditions which exclude heat leakage into the wall (by some such method as Mr. Hering's, described in our December issue) and then to determine afterwards the effect of leakage for various furnace designs and electrode construction.

The Situation in the Iron and Steel Industry

The iron and steel industry has closed a spectacular year. The trade movement has shown remarkable swings as to prices and as to tonnage. With three years of heavy demand and gradually advancing prices, followed by a 15 months' campaign to maintain prices of finished steel products in the face of greatly reduced demand, the market broke Feb. 18, 1909. Perhaps it is more accurate to say that on that date the decision was reached by officials of the United States Steel Corporation, in conference with representatives of some of the important independent interests, to throw the market open, and that thereafter different portions of the price structure collapsed one at a time. Within two days of the decision steel bars suffered a reduction of $4 a ton and plates and shapes a reduction of $6 a ton. One business day later, Feb. 22, steel pipe was reduced five points, or about $9.50 per ton, and two days thereafter sheets dropped $4 to $5 a ton. On March 15 tin plate was cut 25 cents per box. Wire prices were moderately well held to give jobbers an opportunity to distribute their stocks, and on May 1 cuts were made of $7 a ton on nails, $8 on plain wire and $10 on barb wire. In steel pipe, tin plates and wire products the initial cuts were found sufficient to meet the situation, while in bars, plates, shapes and sheets the first cuts were followed by shading which established new low points. There was no cut in standard sections of rails. The subject was canvassed at the time the market on other finished steel products was thrown open, and it was concluded that no large business would follow a reduction. In succeeding months individual manufacturers stood ready to make a sharp cut in rails, if it appeared that large orders could be secured thereby, but it was evident that the railroads were not prepared to buy rails heavily. In no line did it require a long time to develop a bottom. Bars, plates and shapes were trending upward within a little over two months from the time of the original cut; tin plates and pipe did not advance until long after their reductions, but in each case there was but one reduction, the market being held firm thereafter. The lowest prices on wire products were open for a single fortnight, the reductions being made May 1 and the first advances May 15. Sheet products were slowest to find a stable basis, but there was double delay in that case; prices of sheets were reduced Feb. 24, but there was a long delay before the steel mills made their lowest prices on sheet bars; when the minimum on sheet bars was finally reached the independent sheet mills loaded up and fresh cutting occurred in their finished product, and it was not until July that sheet prices showed a discernible tendency to advance.

* * *

Of the wisdom or unwisdom first of the price-maintenance movement and then of its abandonment, *quot homines, tot sententiae*. The game was played with such pomp and circumstance that the sudden abandonment could not but shock the aesthetic senses of the onlookers. Those who had been nearer the stage had heard the creaking of the machinery and had felt that the play was being carried on partly to amuse the audience and partly to keep the players out of mischief. If the undertaking was as serious as its sponsors maintained during its progress, the disruption of the forces should have been regarded as a calamity, but the event has proved what was maintained by some observers at the time, that a definite break would be followed by a rapid recovery. Price maintenance, as has been pointed out before, was no new thing in the steel industry; on many occasions there had been efforts in that direction. The trappings, however, were new. The break has proved to be but

an incident in history. Substantially the entire reduction in finished steel products was recovered by the subsequent advances, except in the case of pipe. Most finished steel products closed the year at prices $1 to $2 a ton below the opening prices, but the closing prices were firmly maintained, while the opening prices were largely nominal, and subject to shading. Merchant steel pipe, however, is a conspicuous exception, as it experienced but a small recovery.

* * *

No less spectacular than the movement in prices has been the movement in tonnage. It strained the capacity of existing plants to make 23,000,000 tons of pig iron in 1905; next year under similar strain the increased capacity produced 25,300,000 tons; in 1907 the industry, still pushed with increased capacity, was making for an output of about 27,350,000 tons and had reached a rate closely approximating 28,000,000 tons in October, but the panic of that month caused a sudden slump and the calendar year's output was only 25,781,361 tons. In January, 1908, the rate of output had dropped below 13,000,000 tons, yet at that time there were new furnaces in course of erection with an annual capacity of 3,500,000 tons. Where was the hope that capacity would again be fully employed, in any reasonable period of time? The year 1908 gave no answer, for its total output fell below 16,000,000 tons, but the year 1909 showed the wonderful resources of this prince-and-pauper industry. It opened with a rate of 21,500,000 tons, showed an average rate of 22,000,000 tons during the first half, and closed with a rate of nearly 32,000,000 tons. With the present rate of production the physical capacity has been closely approximated. There are a number of old furnaces not yet in blast, and several new ones shortly to be completed; apparently there is perhaps 2,000,000 tons of capacity left to come in, but practically the present rate is a fictitious one. Many blast furnaces were relined and otherwise repaired during the depression, and the efficiency of the furnaces as a whole is at present abnormally high. It is normal for close to 10 per cent of the furnaces to be out from one cause or another, but at the moment the idle capacity, including furnaces not completed, is less than 10 per cent of the capacity in blast. It is improbable that the existing plant could steadily make 32,000,000 tons a year. With the new furnaces, some nearly completed and some only projected, a capacity of 33,000,000 tons or more is easily in sight.

* * *

We have been dealing with pig iron. Of the 25,781,361 tons made in 1907, the record calendar year, about 18,000,000 tons went into steel. The gain in pig iron, from a maximum rate in 1907 of nearly 28,000,000 tons a year to a rate of nearly 32,000,000 tons a year at the close of 1909, was almost wholly in steel-making iron. The foundry trade lagged behind in the industrial revival, as it usually does, the trade dictum being that the foundry industry is six months behind the steel industry in such cases. That practically the entire gain should be in the steel industry is remarkable, as the percentage of gain is increased, but it is still more remarkable to observe that rail production has fallen behind, and consumption of plates and shapes in the steel-car industry has certainly made no gain over 1907. The gain in the total, plus an amount to cover these defections, must be found in other lines of consumption. It does not seem that there has been a gain in bridges and buildings. There must be an enormous gain, on the other hand, in the light lines, pipe, wire, sheet products, merchant mill products, etc.

* * *

When the steel industry of the United States confined itself almost exclusively to the Bessemer process, pig iron production was a close index to finished steel production. A given tonnage of Bessemer pig iron being produced, the metalloids were eliminated and a little iron lost, most of the scrap made in casting and rolling went back into the converter, and the finished Bessemer steel was regularly a small percentage below the original pig iron production. Nowadays the case is complicated by the open-hearth furnace, with its scrap and ore additions, some of the scrap being old material but the bulk of it new material produced in Bessemer and open-hearth steel works. We know that in 1909 practically the same tonnage of pig iron was produced as in 1907, and that there was considerably more steel-making pig iron and considerably less foundry iron and other grades. We know, too, that the Bessemer steel works did not do as well in 1909 as in 1907, and hence we are perfectly safe in concluding that, the acid open-hearth being a minor issue, the basic open-hearth furnaces consumed a great deal more basic pig iron in 1909 than they did in 1907, but what good does that conclusion do us as an index to steel production? In past years the production of basic open-hearth steel ingots has been roughly double the production of basic pig iron. If each additional ton of pig iron in 1909 was used with an additional ton of scrap, the gain in steel production was very large, especially since there was less new scrap to be drawn from Bessemer plants. Old material, however, was scarce in 1909, as shown by market prices compared with pig iron, and it may be that some of the additional pig iron production was for the purpose of making up for the shortage in scrap, in which case we should find a much smaller gain in steel output. The official statistics will be awaited with unusual interest. Perhaps the best that can be said at this time is: In 1907 we made 25,781,361 tons of pig iron, the calendar year record, and in 1909 we made substantially the same tonnage; in 1906 we made 12,275,830 tons of Bessemer steel ingots and castings, the record; in 1907, 11,667,549 tons; in 1909, probably considerably less than 11,000,000 tons; in 1907 we made 11,549,736 tons of open-hearth steel ingots and castings, the record; in 1909, probably more than 13,000,000 tons; in 1907 we made 17,664,736 tons of finished rolled steel, the record; in 1909, probably half a million tons either way from 19,000,000 tons.

* * *

The pinch of an insufficient scrap supply noted in the foregoing has not been without its effect in the selection of processes for the future. Late in the year the Bethlehem Steel Company, having before it alternative propositions of adding to its Saucon plant of ten 50-ton basic open-hearth furnaces either 10 similar furnaces or two 20-ton Bessemer vessels for duplexing, chose the latter; the Jones & Laughlin Steel Company experimented with duplexing by combining its Bessemer and open-hearth equipment, originally built with no such idea, and the United States Steel Corporation, which originally made authorizations for the Gary plant, involving an open-hearth ingot capacity more than double the blast-furnace capacity, has been considering duplexing and the addition of blast furnaces without corresponding open-hearth additions.

Perkin Medal.

At the meeting of the Perkin medal committee on Monday, Dec. 13, at the Chemists' Club in New York City, in which delegates of the Society of Chemical Industry, the American Chemical Society and the American Electrochemical Society participated, the Perkin medal for 1910 was awarded unanimously to Dr. Edward G. Acheson, of Niagara Falls. The formal award will be made at the January meeting of the New York Section of the Society of Chemical Industry.

Boston Meeting of the American Chemical Society.

The forty-first meeting of the American Chemical Society is to be held in Boston in the last week of December. It will begin on Monday, Dec. 27, and last to Friday, Dec. 31. Special arrangements have been made for the entertainment of the visitors, while the program for professional papers to be presented before the different Sections contains not less than 111 papers. Of these there are 60 to be presented before the Division of Industrial Chemists (Mr. A. D. Little, chairman). Before the same section there will also be presented a symposium of papers on the chemistry of paints.

Of lectures of general interest we note the report by Dr. F. W. Clark for the International Committee of Atomic Weights, a paper by Dr. T. W. Richards on methods employed in precise chemical investigations, a paper by Dr. Louis Kahlenberg on the past and future of the study of solutions, and the address of the retiring president of the American Chemical Society, Dr. W. R. Whitney, on "Some Chemistry of Artificial Light."

Dr. C. L. Parsons, Durham, N. H., is the secretary of the society.

Annual Meeting of the Western Association of Technical Chemists and Metallurgists.

The fifth annual meeting of the Western Association of Technical Chemists and Metallurgists will be held at Boulder, Golden and Denver on January 6, 7 and 8, 1910. There is an interesting program of papers and a number of visits have been arranged by the local committees to various industrial plants, including smelters, beet sugar factories and breweries. The various laboratories of the State University and the State School of Mines will also be visited.

Mr. H. C. Parmelee, 1510 Court Place, Denver, Colo., is the secretary of the association.

Philadelphia Meeting of the American Institute of Chemical Engineers.

The second annual meeting of the American Institute of Chemical Engineers was held in Philadelphia from Dec. 8 to 10 and the attractive program, which was printed in our last issue, was successfully carried out. The attendance at the meetings and numerous excursions and at the dinner in the Hotel Walton was very satisfactory.

The address of the retiring president, Dr. S. P. Sadtler, on the relation of the chemical engineer to the conservation problem is printed in full on page 9 of this issue, while abstracts of some of the papers presented will be found in the Synopsis in this issue. Of other papers presented we notice one by Mr. E. R. Taylor on the electric furnace for smelting iron ore, which covered in a new form essentially the same ground as his Electrochemical Society paper published in our last issue.

The following officers were elected for next year: President, Dr. Chas. F. McKenna; vice-presidents, Dr. F. W. Frerichs, Dr. Edward G. Acheson and Dr. Eugene Haanel; secretary, Dr. John C. Olsen, Polytechnic Institute, Brooklyn, N. Y.; treasurer, Wm. M. Booth; auditor, H. D. Renaud; directors, G. B. Adamson, D. Wesson. Dr. E. Gudeman, L. Reuter, Thorn Smith, H. F. Brown, Dr. W. M. Grosvenor, R. K. Meade, Dr. S. P. Sadtler.

Dr. Chas. F. McKenna, the newly elected president of the American Institute of Chemical Engineers, is a new Yorker by birth, 48 years of age, an M.A. of St. Francis Xavier's College and a Ph.D. of Columbia University. He was chairman of the committee of six which organized the Institute and has been first vice-president since its organization.

His earliest work in chemistry was with the Havemeyer Sugar Refining Company and the Cambria Steel Company.

In 1887 he designed, erected and operated a plant for the manufacture of lime for the purification of gas, and this led him first into the study of the utilization of lime wastes and later into the field of limes, mortars and cements, in which he has won a high reputation. In the manufacture of Portland cement he has acted for many of the manufacturers in the study of their raw materials and design of plants.

In the beginning of his work as a consultant and chemical expert he was associated with Dr. Gidson E. Moore and established a testing laboratory of which he is still the head and which has aided him to attain a position of authority upon materials their qualities and processes of treatment.

He has been a chemist to several metallurgical and manufacturing concerns and is frequently consulted by the authorities of his own city, while his devotions to the interests of chemistry, chemists and chemical education are well known and generally recognized.

The New Chemists' Club Building.

With great pleasure we are able to state that the preliminary arrangements for the erection of a new large Chemists' Building, worthy of the industry which it is to represent, have been successfully concluded.

The capital of the Chemists' Building Company is to be $250,000, of which $210,000 have been subscribed by nearly 100 individuals or companies. The remainder of the money for the erection of the building will be raised by a mortgage.

The directors of the Chemists' Building Company are: Dr. Morris Loeb, president; Dr. Chas. F. Chandler, vice-president; Albert Plant, treasurer; W. H. Nichols, Jr., secretary; Dr. Leo H. Baekeland.

As already stated, the building is to be erected at 50 to 54 East Forty-first Street, a plot 56 x 100 ft.

The six lowest stories will be devoted to the purposes of the Chemists' Club, while the upper five floors will be used for laboratories.

At the entrance to the club rooms will be a reception room, office and coat room, back of which is a large hall, 50 ft. square, which runs over two stories. The front of the second story will be devoted to billiard room, etc. The third story will contain a large social room and the restaurant. The front of the fourth story will be devoted to the library, and the back to the board room, a dark room and a chemical museum. On each of the next two floors will be found 10 bedrooms for club members.

It is hoped that the library will be made a thoroughly complete one, and house not only the books of the Chemists' Club, but those of the American Chemical Society and those of the Perkins Fund as well.

The Museum will be devoted to the collection of articles of scientific interest, rather than such as are adapted to popular exhibitions. It is also hoped that it will contain specimens of all kinds of chemicals to be used for research, in about the same way as a circulating library; that is to say, specimens can be taken out for experiments, or for comparison, on the condition that they be returned or replaced within a given period. The meeting room will be let out to scientific societies and others.

The laboratory floors are to be provided with all the mains necessary to supply the individual laboratories with gas, electricity, water, etc. They are also to contain ventilating flues,

and such provision for hoods that these can be introduced at minimum cost. Indeed, they could probably be provided at once to any tenant who made a lease before the completion of the building. These facilities have been so designed as to make a subdivision of each floor into four individual laboratories with adjacent offices, but, at the same time, it will be easy for a single tenant to make satisfactory use of a whole floor, or half a one.

While the rentals of the laboratories must, necessarily, provide an income for the building, they will be placed at as low a figure as possible, in order that commercial chemists as well as scientific investigators may obtain laboratories at a reasonable cost under far better conditions than those prevalent elsewhere. The advantage will be a fireproof laboratory, with the same improvements that can be found in a modern college building, a central library within reach and those benefits which usually go with the occupancy of a building devoted to a single profession.

Of course, these laboratories can also be rented out for work in cognate sciences, such as physics and biology.

The stockholders have agreed to limit their dividends to 3 percent, and any surplus will be applied to reducing the capital indebtedness of the company. At the same time the Chemists' Club has an option to purchase the entire stock of the company at par within the next 50 years. This will make it possible for the club to purchase the building whenever the capital has been sufficiently reduced by the operation of the provision just stated.

On the other hand, should the club for any reason fail to prosper, the stockholders will have the building, which is always applicable for business purposes, since it is in a central location, and will be constructed in a perfectly satisfactory manner for conversion into ordinary offices.

Municipal Chemistry.

The Department of Chemistry of the College of the City of New York (Prof. Chas. Baskerville, Director) offers during the Spring Semester a course of thirty lectures on the chemistry of daily life.

These lectures will be open to three classes of hearers:

1. Senior students of the college who have complied with the requirements of the department. Laboratory practice (no fees) involving the examination of water, milk, foods, gas, paints and cements, will be had under the direction of Associate Professor Moody.

2. Employees of the city who have studied sufficient chemistry to pursue the laboratory work. The attainments of each applicant will be passed upon individually. The expense involved in apparatus broken and chemicals used must be met by the hearer.

3. A limited number of auditors composed of citizens of New York City will be admitted on applying for a seating to the Director of the department. The last are not admitted in the laboratory part of the course.

The lectures will be given as per printed schedule, at 4 p. m. in the Doremus Lecture Theatre, Chemistry Building, 140th Street and Convent Avenue, Plaza entrance.

All communications should be addressed to the Director, Dr. Chas. Baskerville.

The course of lecture is as follows:

Feb. 4th, Prof. Charles Baskerville, Sanitation.

Feb. 9th, Dr. Wm. P. Mason, Drinking Water and Disease.

Feb. 10th, Dr. Wm. P. Mason, Sources of Municipal Water Supply.

Feb. 11th, Dr. Wm. P. Mason, The Purification of Polluted Water.

Feb. 15th, Dr. Thomas C. Darlington, Milk.

Feb. 18th, Dr. Harvey W. Wiley, The Purpose, Method, and Extent of Food Adulteration.

Feb. 19th, Dr. Harvey W. Wiley, The Remedy of Food Adulteration and Relation of Chemistry thereto.

Feb. 25th, Mr. Bayard C. Fuller, Food Inspection.

March 1st, Mr. Bayard C. Fuller, Spoiled Foods.

March 4th, Dr. Virgil Coblentz, Drugs and their Adulteration.

March 8th, Dr. Virgil Coblentz, Methods for Detecting Adulteration.

March 11th, Dr. Virgil Coblentz, Habit Inducing Drugs.

April 1st, Dr. Allerton S. Cushman, Streets and their Construction.

April 2d, Dr. Allerton S. Cushman, Modern Road Construction.

April 5th, Hon. William H. Edwards, Street Sanitation.

April 8th, Mr. Edward D. Very, Disposal of Ashes and Light Rubbish.

April 12th, Dr. Edward D. Very, Disposal of Garbage.

April 15th, Dr. Edward D. Very, Disposal of Putrescible Materials.

April 19th, Dr. Arthur H. Elliott, Manufacture of Gas.

April 22d, Dr. Arthur H. Elliott, Means of Testing the Properties and Quality of Gas.

April 26th, Dr. Arthur H. Elliott, The Smoke Problem.

April 29th, Dr. Herbert R. Moody, Ventilation.

May 3d, Dr. Thomas A. Storey, The Chemistry of Personal Hygiene.

May 10th, Mr. Maximilian Toch, Paint and Painting.

May 13th, Mr. Maximilian Toch, Corrosion of Metals and its Prevention.

May 17th, Mr. Maximilian Toch, Cement and Concrete.

May 20th, Dr. A. A. Breneman, Combustibles and the Causes of Fires.

May 24th, Dr. A. A. Breneman, Methods of Extinguishing Fires.

May 27th, Dr. N. L. Britton, City Parks, Gardens, and Playgrounds.

The Iron and Steel Market.

December has been a very quiet month in the iron and steel trade, but the market has shown a firm undertone and the lull is regarded as merely the seasonable one which is practically inevitable at this time. No sudden resumption of activity is expected, but a gradual resumption of buying is anticipated as the first quarter is well entered upon, and producers are holding prices of finished steel products firmly, with the tendency to ask higher prices for first half than for first quarter deliveries.

The high pace of production reached in November continued through December. No important additions to capacity were made in the month outside of the blowing in of a few blast furnaces here and there. The iron and steel industry had gotten up very close to the limit of its capacity in October and November, and looked rather for some decrease in output in December on account of weather conditions, but these have been fairly favorable, and the month's statistics are likely to show the best output of the year.

Price changes in December were unimportant, the principal finished steel products being untouched, with the exception that at the close of business Saturday, December 12, a general advance of $1 a ton was made in wire products. These prices apply to jobbers, with whom it is customary to make contracts requiring specifications within 60 days. Specifications on contracts placed at previous prices were not heavy, and the advance was presumably made chiefly with a view of stimulating specifications. Prices had already been made to manufacturers using plain wire at a $2 advance on contracts for deliveries over the first six months of the new year, so that the advanced price to jobbers is $1 a ton below the price made to manufacturers, and another $1 advance to jobbers is expected January 1 or January 15.

Specifications have been heavy on contracts for finished steel products expiring December 31, this applying particularly to steel bar contracts. Such contracts carried prices below the present level, and buyers were anxious to escape having any of the contract tonnage cancelled by the mills at the time limit.

The finished steel trade enters the new year with fully three months of actual operations specified on books in the case of steel bars, and nearly as much specified in plates and shapes. Sheets and tin plates are sold under definite contract involving full half, and perhaps more than half, the prospective production in the first half. Wire and pipe are sold less far ahead, as it is contrary to the policy of the mills to sell for more than about 30 to 60 days ahead.

As a result of the disorder attending the attempt to start the Ætna-Standard sheet plant, of Bridgeport, O., of the American Sheet & Tin Plate Company, it seems to have been decided to abandon all efforts to start the Wheeling district plants. June 30, on the expiration of the annual scale, the members of the Amalgamated Association walked out of all the sheet and tin-plate mills of the American Sheet & Tin Plate Company which had hitherto been union, with the exception of the Guernsey sheet plant, at Cambridge, O., the company having announced that it would operate on the open-shop principle thereafter. Subsequently operations were started at all the sheet and tin plate plants affected, with non-union labor, with the exception of the plants in the Wheeling district, comprising the Ætna-Standard sheet plant at Bridgeport, O., 23 mills; the La Belle tin plate plant at Wheeling, 10 mills, and the Laughlin tin plate plant at Martin's Ferry, O., 23 mills. Wheeling has been a stronghold of union sentiment, and no effort to start operations was made until December 1, when a few mills at the Ætna-Standard plant were started with old workmen, who had requested the management to give them employment. This brought about a small riot, with the calling out of 1500 troops. Cecil A. Robinson, manager in the Wheeling district, has resigned, effective January 1, while D. A. Barrett, superintendent of the Laughlin plant, has been transferred to the Morewood tin plate plant at Gas City, Ind., containing eight mills, which have operated only about one month in the past four years. It is intended to put the Morewood plant in operation, while there appears to be no intention of trying to start any of the Wheeling district plants, at least in the near future.

Statements have been made regarding the building by the Steel Corporation of a large sheet and tin plate plant at Gary, Ind., but it is possible that the announcements at this time are merely an incident of the labor difficulty, since the plans for 50 tin plate and 50 sheet mills for Gary were drawn up more than a year ago. Through the dismantling of a number of sheet and tin plate plants by the American Sheet & Tin Plate Company and the building of a large number of independent plants the Steel Corporation has lost much in relative position, and would not entirely restore its former relations in the industry by building even a 100-mill plant at Gary. The growth of tin plate and sheet consumption has been very rapid, and there is no reason to suppose that demand would not be found for the additional output.

Coke.

Asking prices for Connellsville furnace coke have yielded slightly in the month, to $2.75 or $2.80, but not enough to induce the uncovered furnaces to come into the market on contracts for the new year. During the spectacular advance from less than $2 in July to $3 late in September a number of contracts were made, two or three small ones being understood to have been made at the top price, but there remains large uncovered requirements. Consumers' expectations seem to be that there will be no advance, and that there may be a decline. Meanwhile, if they do not make contracts, they can buy prompt coke from time to time.

Pig Iron.

The local pig iron market has been quiet, but prices have been well held. Bessemer is $19, valley, for any delivery, but no important sales have been made since those late in November whereby two independent steel interests took a total of 28,000 tons at this price for the first quarter delivery. Moderate sales of basic have been made for first quarter delivery at $17.25, valley, there being a little tonnage reported as available at $17 for immediate shipment. Foundry iron is $17 to $17.25, valley, depending on delivery, with forge about 75 cents less. Prices delivered at Pittsburg are 90 cents higher than valley. Owing to the offering of speculative Southern iron at $14.50, Birmingham, and then at $14, but only for immediate shipment, the Southern furnaces have yielded from their $15 price and have been ready to do $14.50 for first-half delivery. This price has brought out a moderate amount of business.

Finished Materials.

Prices of finished steel products are very firmly held as below. In cases where a range is indicated, the higher price is for the more extended delivery. Prices are at Pittsburg unless otherwise noted:

Standard steel rails, $28 for Bessemer, $30 for open-hearth, f.o.b. mill, except Colorado.

Plates, $1.55 to $1.60 for tank quality.

Shapes, $1.55 to $1.60 for beams and channels, 15-inch and under, tees, and angles 2 x 3 and larger.

Steel bars, $1.45 to $1.50, base.

Iron bars, $1.70 to $1.75; Chicago, $1.60.

Wire nails, $1.85 base; plain wire, $1.55; galvanized barb wire, $2.15.

Black sheets, 28-gage, $2.40; galvanized, $3.50; blue annealed sheets, 10-gauge, $1.75; painted corrugated roofing, $1.75 per square; galvanized corrugated roofing, $3.

Tin plates, $3.60 for 100-lb. cokes.

The Faraday Society.

The forty-seventh meeting of the Faraday Society was held on Tuesday, November 30, 1909, at the Institution of Electrical Engineers in London. Mr. James Swinburne was elected president.

A paper by Dr. Henry J. H. Sand dealt with the **electro-analytical determination of lead** as peroxide.

This investigation was carried out with the object of elucidating the cause of the discrepancy between the statements of various experimenters regarding the behavior of an electrolytic lead peroxide deposit in drying at 200°. All authors agree that the peroxide deposit retains water at this temperature, but whereas Hollard and Bertriaux give an analytical factor of 0.853, most other investigators find a factor of approximately 0.864, the theoretical factor being 0.866. Incidentally, the effect of varying conditions on the coherence of the deposit was also studied.

It was found that at 200° a lead peroxide precipitate is capable of absorbing moisture from a damp atmosphere, and an increase of as much as 1.7 per cent of the weight of the deposit has thus been obtained. On heating in a dry atmosphere at the same and higher temperatures, the peroxide loses its water exceedingly slowly.

The most satisfactory results were obtained by depositing the peroxide in such a manner that it is practically free from water owing to electric endosmose on leaving the bath. The conditions are a high temperature and a large current density (with a rotating cathode). The deposit was dried by means of alcohol and ether in the identical manner employed for metal precipitates.

A paper by Mr. Arthur Jacques discusses the influence of dissolved **gases** on the **electrode potential** in the system of silver, silver acetate, aq.

Mr. Arthur Jacques also contributed a paper on **ionisation** in aqueous solutions in lead acetate and cadmium acetate.

A paper entitled "The **calorimetrical analysis of hydrated salts**" was communicated by Prof. F. G. Donnan and Dr. G. D. Hope. The authors point out that the interpretation of the heats of solution of hydrated and partially dehydrated salt given by Thomsen in his Thermochemische Untersuchungen is in various cases either erroneous or unsatisfactory.

It is shown that Thomsen's data for sodium carbonate indicate," when correctly interpreted, the existence of only the

hydrates with 1, 7 and 10 mols. water per mol. anhydrous salt. The authors' experiments confirm this result.

In the case of copper sulphate, neither the experiments of Thomsen nor those of the authors indicate more than the existence of the hydrates $CuSO_4 . 5H_2O$ and $CuSO_4 . H_2O$, though the hydrate $CuSO_4 . 3H_2O$ is known to exist.

This calorimetric method may be employed as an aid in the discovery of the existence of salt hydrates, though it is probably neither so easy nor so reliable as other well-known methods.

The Aluminium Situation.

According to a communication published from trade circles in the *Frankfurter Zeitung* and noticed in the London *Electrical Review*, the hopes of an understanding among the French producers of aluminium, after months of preparatory work, had recently to be abandoned. The inland understanding between the French firms, who produce a very large part of the total output of aluminium, would naturally have formed the basis for a far-reaching combination. But from the beginning the allotment question or the regulation of the output of the individual works seems to have been incapable of solution.

It is quite obvious that the powerful undertakings which during the period of high prosperity carried out what were doubtless exaggerated extensions now object to renounce their comprehensive schemes in relation to production. Nevertheless, they will find themselves, if they also wish to improve the price of the metal, in the situation which they have themselves created through the excessive straining of the (late) syndicate prices.

The price of aluminium improved from the time of the appearance of the first syndicate rumor in June, 1908, but since the recent definite abandonment of this community of interests consumers have manifested a natural reserve. This circumstance is aggravated by the accumulating offers of intermediaries who seem, owing to the continued circulation of syndicate rumors, to have supplied themselves with large quantities at the lower prices prevailing until recently. But as their speculative purchases were not so quickly realized as they had hoped for, they are inundating the market with cheap prices, which again appear at about 1s. 3d. per kilogram (14 cents per lb.) and lower for crude aluminium.

Apart from this, consumers are deriving advantage from the situation. There is no doubt that the consumption of the metal is increasing and must in all probability continue to progress if the price is kept within normal limits. These may be regarded as from 1s. 5d. to 1s. 7½d. per kilogram (16 to 18 cents per lb.) for crude ingots or plates with an extra of from 5d. to 5½d. (10 to 11 cents) for the rolling out of sheets. On the basis of this charge for sheets, the manufacture of aluminium vessels and various articles is remunerative, but the overstepping of this maximum would reduce the consumption. In conclusion, it is stated that the present average price of 1s. 3d. per kilogram (14 cents per lb.) for crude aluminium only represents a small profit, but it no longer directly results in a loss.

All this refers, of course, to Europe. For this country the protective tariff is to be taken into account.

CORRESPONDENCE.

Electrical Smelting of Copper Ores.

To the Editor of Metallurgical and Chemical Engineering:

SIR—The electric furnace has acquired a footing in the iron and steel industry, but as yet little has been done in the field of copper smelting. As an accessory to a matte-smelting blast furnace or reverberatory, electric smelting is, however, capable of being employed to advantage in many cases.

As a general proposition, the fuel energy required to increase the temperature of one unit of mass by one unit of temperature increases greatly with the initial temperature of the mass, and to raise the temperature of a given mass from 1500 to 1600 deg. requires more fuel energy than is required to heat it from 100 to 1100 deg., and vastly more than is required to heat it from 500 to 600 deg. In general it increases in a geometrical progression as the temperature increases. Heat generated in a mass by electrical resistance is, on the other hand, cumulative, and except for loss by conduction and radiation it takes no more current and costs no more for 100 deg. increase of heat at 1500 deg. initial temperature than at 1000 or 500 deg.

This points to a field where electrical heat can be used to advantage, viz., for carrying the summit of the smelting load under conditions where fuel heat does the work of smelting up to the point where an electric current can act upon the material to generate internal heat, the electric current then finishing the operation. In other words, to an adjustment of conditions such that the smelting by fuel heat will proceed to the point where there will be formed a semi-fluid mass capable of flowing and forming a bath, when the completion of the smelting will be done by electric heat.

In the smelting of sulphide ores we are dealing with material of a complex nature, composed of elements or constituents having different temperatures of fusion, and the constituent of lowest smelting temperature fuses first and the others in the order of fusibility.

As in the freezing of molten alloys and mixtures where the element of highest freezing temperature solidifies first and the elements of lower freezing temperature remain as land-locked bodies within the chilled mass until the temperature falls to their respective freezing points, so in the fusion or smelting of ores containing components having different fusion temperatures, the constituents of lowest smelting temperature fuse first and drain down into the bath, carrying downward particles of unfused constituents of a higher fusion temperature or leaving a spongy mass composed of such constituents. In straight fuel smelting in a blast furnace conditions require that the fusion of all elements, those having a high-fusion temperature as well as those of a low smelting point, be accomplished at or above the plane of the tuyeres. Below that plane the space is filled with incandescent coke, which, being in a non-oxydizing atmosphere, remains as a support for the charge. At the plane of the tuyeres everything except the coke melts and drains down through the incandescent coke bed into the bath. The result is that in straight fuel smelting the consumption of coke is largely in excess of theoretical requirements, for it must be sufficient to make a free-flowing fluid of all material and preserve the coke foundation. In straight pyritic smelting the problem appears to be complicated by the small margin between the fuel energy of the contained sulphur and iron and the heat requirements of complete fusion, and further, probably, by the formation of strata of material of high fusion temperature below the tuyere plane in lieu of the incandescent coke foundation before noted.

The electrical heating of the bath in a matte smelter will permit of the cutting down of the coke charge to the point where the coke will do the smelting of the more fusible material and the electric current the balance, or if the fuel value of the contained sulphur and iron approximates to that required for the basic smelting, the use of coke can be dispensed with. This can be secured by a hearth of a trough character with electrode terminals, and also a fore hearth with a flowing tap, so as to maintain in the trough hearth a bath of constant level or cross-section. The hearth should so discharge into the forehearth that there will not be an accumulation of matte. The current, of course, is of large amperage and low voltage, the resistance of copper slag indicating a working voltage of from 5 to 7 volts. Estimates indicate that 1 electric horsepower will give a control of 200 deg. (Fahr.) per minute per ton of ore capacity per day, so that 1 electric horsepower will heat 1.4 pounds matte-slag 200 deg. per minute.

This should suffice, for it is to be borne in mind that the elec-

tric heat is cumulative, and is added to material already close to the fusion point; that the time or capacity element is the one to be affected by the quality of the current; that if the volume of current is small it simply takes a longer time for it to supply the deficiency for smelting a given quantity of semi-fluid material than it will take when the volume of current is larger. In other words, that the electric current is a governor supplying just what heat is needed for each unit of product to give a uniform temperature.

With the coke charge reduced to the amount simply required for the basic fusion and all consumed at or above the tuyeres, or in the case of pyritic ores eliminated entirely, the result will be that the space below the tuyere plane will be occupied by material of an open, spongy character of progressively varying fusibility with that of highest fusibility at the bottom. The fused products of the fuel smelting draining down into the bath will, with the excess of heat they carry down, complete the smelting of matter of the lowest smelting temperature met with.

Elements of lowest fusibility will be smelted out in advance of elements of higher fusibility, and consequently the material resting on the bath and pressed down into the bath and extending up to the plane of the tuyeres will be of varying fusibility with that of highest fusibility at the bottom. The electric heat is therefore expended directly on the material of highest fusibility, while fuel smelting starts with the material of lowest fusibility and works up in the scale of temperatures, so to speak, to the elements of higher fusibility. Under proper adjustment of current and fuel, fuel smelting and electric smelting will meet at the point where, everything considered, the result is most economical.

If accessory electric smelting is to be used as a control for pyritic smelting so as to make it practical, the current need not necessarily be so very heavy, for, being cumulative, the electric heat will in time supply the balance required.

Naturally, we look to hydro-electric power, where it is obtainable, as the source of current supply. But accessory electric smelting will probably permit of the use in many cases of comparatively cheap fuels, oil, producer gas, waste, heat, etc., in lieu of high-priced coke.

Electric smelting can not only be employed in connection with blast-furnace smelting, but it can be used in reverberatories. The adaptation to reverberatory practice of electric smelting as above outlined calls for a new type of reverberatory, one in which there can be maintained two long banks of ore or charge with a trough between, this trough having electrode pockets or connections at both ends and a forehearth with a flowing tap. The reverberatory has rows of charging holes in the roof next to the sidewalls and the charge, banked against the sidewalls and protecting them, presents a crest of summits and saddles with slopes leading down to the central trough into which the semi-fused material drains.

In a reverberatory furnace a large amount of waste heat is available for current generation, and thus the entire heat energy can be utilized, either primarily as fuel heat or secondarily as electric heat.

Story B. Ladd.

Washington, D. C.

Electric Smelting of Iron Ores at Domnarfvet and at Nykroppa, Sweden.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—In order to show to what extent the Iron & Steel Association of Sweden are putting faith and confidence in the work and results already accomplished in electric smelting of iron ores by the use of the "Elektrometall" Company's electric smelting furnace, it is interesting to learn that the "Jernkontoret," at a meeting in November last, appropriated the sum of 300,000 krone toward the immediate construction of an experimental electric furnace plant at Trollhättan, of the above type and of 25 tons daily capacity.*

As already has been mentioned in the *Electrochemical and Metallurgical Industry*, a 700-hp electric furnace of this type was in successful operation at Domnarfvet for some time, and, according to the very complete report† of Lars Yngström in the transactions of the "Jernkontoret" (Part 9, 1909), these results have demonstrated that the question of electric production of iron from ore "is now economically as well as practically solved."

The furnace which the Swedish Iron & Steel Association is to erect, therefore, ought not to be considered as a further experiment in this direction, but as an "experimental plant" to which any iron manufacturer or mine owner can send his ore and have the same treated for obtaining information as to its suitability for such a process, the purpose for which the resulting metal would be best adapted, and the cost of manufacture, as compared with iron produced from the same by means of charcoal or coke.

The business arrangements arrived at between "Jernkontoret" and the "Elektrometall" are reported in the Swedish daily press to comprise the following principal parts:

The "Elektrometall" shall without further cost execute and furnish all necessary detailed drawings and specifications of building and furnace appliances; also during the construction period furnish all required advice and information.

The "Jernkontoret" shall defray all expenses in connection with the construction of the plant and have full charge and final decision as regards the same. The Jernkontoret reserves the right of exercising the patent rights for their own use or to dispose of such rights to others, in consideration of a first payment equal to 2 krone per electric horse-power for each particular size furnace or furnace type erected, besides a royalty of 1.25 krone per metric ton pig iron produced according to the "Elektrometall" Company's patents during the life of the same.

From this, however, is excepted the present experimental plant, for which no further charge or royalty shall be paid.

As regards power for the present plant, the "Jernkontoret" has made an arrangement with the Swedish "Hydro-Power Commission" whereby during two years they will, for this particular purpose, obtain up to 3000 hp at Trollhättan for the annual sum of 10,000 krone (equal to about $3) per horse-power year.

In the petition to the Government the "Jernkontoret" gives some interesting information on the results obtained from the late electric smelting experiments at Domnarfvet and Nykroppa: that during said experiments charges of different kinds of iron ores and briquettes have been used (both high and low in P-contents); that the coal consumption has been brought down to about one-third of that of the ordinary blast furnace practice; that the maximum efficiency of the furnace so far has been only 58 per cent (whereas that of the ordinary blast furnace can be considered to be about 80 per cent), but that there is good reason to expect a still further reduction in heat losses for the furnace proposed, by changes in the water coolers about the electrodes and in adopting thicker furnace walls. Mention is also made of the superior quality of some of the electric pig iron made from the Tuolluvara ore at Nykroppa and used in one of their 10-ton O. H. furnaces. This was white iron, which usually is cold and sluggish, but which here proved to be the very reverse, the iron being hot and liquid, and allowing even a higher percentage of ore charge than the ordinary. The cold-drawn pipes made from the steel products proved also to be of superior quality.

Owing to the successful results obtained from the Swedish experiments a large Norwegian electric smelting plant has also been decided on, to be constructed in accordance with the principles embodied in the "Elektrometall" Company's patents.

E. A. Sjostedt.

Sault Ste. Marie, Ont., Canada.

*This was already briefly noticed in an editorial note on page 503 of our December issue.—Ed.

†A translation of this report will be found on page 11 of this issue.—Ed.

Conservation and the Chemical Engineer.*

BY SAM. P. SADTLER.

We have heard much in the last year or two concerning the Conservation of our Natural Resources and we shall, I feel certain, hear much more in the next few years, as the facts elicited from the preliminary studies of the subject come to be understood by the public at large. The importance of the subject will grow correspondingly as the matter is studied by the thoughtful citizen and his appreciation of it will in time be reflected in the activity of the statesmen at Washington in the proposing of remedial measures.

Conservation let us note, however, represents the third stage in the history of the development of Natural Resources.

The first stage is Exploration or discovery. This is the era of the prospector and has given us in this country some famous episodes. We need only recall the discovery of gold in California in 1849 and the way in which it operated to attract adventurous spirits from the older parts of our country, or the repetition of the same story with the discovery of the rich gold deposits on the Yukon and at Nome in Alaska.

The first discovery of rich petroleum deposits of Western Pennsylvania in the early sixties brought, similarly, multitudes of prospectors or "wildcatters," as they came to be known locally, and this experience has been repeated also from time to time as great petroleum gushers or powerful gas wells have been reported in various sections of the country, resulting in the opening of new fields, as in Ohio, Indiana, Kansas, Texas and Oklahoma.

The second stage is Exploitation, when these lavish gifts of nature are worked with a view mainly of increasing production and usually in a wasteful way with no thought of the exhaustion of the supply.

As illustrations of this stage we need only cite the way in which our coal mines have been worked. In Bulletin 394 of the U. S. Geological Survey (Papers on the Conservation of Mineral Resources) we find the statement that "it has been estimated that the actual loss or waste sustained through coal left in the mines in conducting mining operations amounts to 50 per cent of the quantity produced and marketed." Worse than this, when the Anthracite Coal Waste Commission made its report in 1893 they estimated that "for every ton produced 1½ tons were lost."

One of our most valuable gifts of nature is the natural gas, which is associated more or less directly with petroleum. It is a fuel of the greatest value, being nearly pure hydrocarbon in its composition. Yet we find in the same Bulletin of the Geological Survey before referred to the following: "As to the amount of natural gas which is being wasted daily, no accurate statistics have been attempted and the judgment of Dr. I. C. White, State Geologist of West Virginia, may well be accepted to the effect that no less than 1,000,000,000 cubic feet of gas are wasted every twenty-four hours. Of this, undoubtedly the larger part is wasted in the production of oil." This waste, Dr. Day of the Geological Survey says, "practically equals the annual consumption of natural gas reported for 1907. This waste should furnish light for half the urban population of the United States."

The exploitation of our once great timber resources and the resulting denudation of great tracts of land and the evils to the soil which have followed in the train of this ruthless waste have been so graphically portrayed by the U. S. Forestry Bureau and others recently that I need not more than allude to this case of reckless extravagance.

It is not my intention, however, to take up this evening the question of the conservation of our natural resources and the absolute need thereof, as so ably developed in recent reports of the "National Conservation Commission," the Forestry Bureau, and other official publications, nor yet the part played by the chemist in this Conservation of Natural Resources which has been so ably reviewed by Dr. Bogert in his recent Presidential address before the American Chemical Society.

What I would like to do is to indicate that these same three stages of exploration or discovery, exploitation or effort at production, and finally conservation, are to be seen in the history of every great chemical industry and to point out, that, while it is the part of the chemical engineer to aid in the exploitation step, what is still more important is his part in counselling and indicating how the wholesome influence of conservation can be applied, so as to broaden and extend the scope of the industry, to maintain and add to its remunerative character and to give it stability and promise of permanence.

Let us illustrate this view by examples and take first the great petroleum industry to which reference has already been made in speaking of the era of exploration when the petroleum fields were first outlined by the work of the prospector and driller. The second stage of exploitation began to draw upon the help of those who were chemical engineers in fact if not by title. The development of the distilling and refining processes came first. Starting with the old cheese-box still with its circle of heating grates under it, there was a change, following upon the discovery of the "cracking process" as applied to crude oil, to the present form of cylindrical still with its movable cover to regulate the chilling of the vapors during the latter part of the operation. This simple feature in distilling enables the refiners to get 75 per cent or more of burning oil from the crude petroleum instead of the 45 per cent of a normal fractional distillation, while producing a residuum which can be advantageously distilled for paraffin oils.

The proper control of the acid and alkali treatment of the crude distillates, the change from the old forms of presses for paraffin scale to the modern filter press, the introduction of bone-black filtration for reduced oils and residuums all contributed to develop and expand enormously the several parts of the industry to which they were applied. With these improvements in large scale methods went the inauguration of improved testing and analytical methods, so that uniformity of product so essential for sound business development was secured.

Let us turn now to the newer evidence of the work of the chemical engineer in the way of conservation as illustrated in this same industry. The collection and utilization for fuel purposes of the uncondensed gas from the distillation of crude oil is one of the important economies that has been generally adopted. The working up of the sludge-acid and recovery of not only sulphuric acid but of valuable side products is now a feature of the larger refineries. The introduction of clay filtration to improve the quality of the heavier grades of oils is an important advance as well as a step of economy. The production of gas oils from the less valuable crude oils has also become an important industry as these are of great value for gas-making and gas-enriching. The more thorough utilization of residues is also a feature of recent years. From these residues are now made excellent road oils for the aid of the good roads movement. The petroleum pitch is all utilized also partly for electric light carbons and partly for fuel purposes. Most promising of all, however, is the result, not as yet fully attained, but most certain of solution in the immediate future, viz., the utilization of crude petroleum of the lesser valuable kinds and residuums in internal combustion engines for the development of power.

A second typical industry is that of coal distillation. We have already spoken of the wastage in the mining of coal. It is not necessary here to speak of the corresponding waste in its utilization as fuel. We will speak solely of the distillation of coal. This treatment may be carried dut from either one of two points of view, and the exploitation in each case has been pushed with great energy, utilizing all available chemical and engineering skill. The first is the distillation for the manufacture of illuminating gas. While the mechanical side of this process has advanced steadily, particularly after the introduction of regenerative firing and mechanical stoking, the chemical side did not

*Presidential address delivered at the second annual meeting of the American Institute of Chemical Engineers in Philadelphia, Dec. 8, 1909.

advance so rapidly. While the coal-tar is no longer thrown away, unless it be in very isolated localities, the proper chemical utilization of this tar has lagged behind both in this country and in England. On the other hand, the true conservation of this valuable side-product and the development of its possibilities has advanced in a notable degree in Germany, the home of the coal-tar industry. Here the research chemist and the chemical engineer have gone hand in hand in the building up of a great industry or rather two industries based upon the utilization of the coal-tar, the manufacture of the coal-tar dye colors, and the manufacture of synthetic medical preparations possessing valuable therapeutic characteristics. Besides these most important and highly developed illustrations of conservation, we have, however, some minor utilizations of coal-tar or products from the same that deserve mention. Thus the manufacture of creosote oils for the preservative treatment of wood, the roofing-pitch and tar-paper manufacture and the use of pitch in the briquetting of coal are illustrations of value given to the coal-tar and its products.

The thorough extraction of the ammonia and the impurities like the cyanides from the ammonical liquor and the production of commercially valuable products from them is, moreover, an accomplishment in the way of conservation. The utilization, too, of the gas carbon for the manufacture of electrode carbons, battery plates and for electric light carbons is another illustration of this work.

The second method of distilling coal is that for the production of coke for metallurgical purposes. This has been developed or more properly exploited to such a degree that, according to the "Mineral Resources of the United States," published by the U. S. Geological Survey, the production of coke for the year 1907 was 40,779,564 tons, of which, however, 35,171,665 tons were produced in beehive ovens. In these, as is well known, only the fixed carbon of the bituminous coal is saved and all volatile constituents including gas, tar and ammonia are absolutely wasted. On the other hand, 5,607,899 tons of coke were produced in by-product recovery ovens and the value of the by-products (gas, tar and ammonia) obtained therefrom amounted to $7,548,071. It is easy to reckon from this what the loss was on the 35,171,665 tons of coke made in beehive ovens. In fact, the article on "Coal," in Bulletin 394 before referred to, says with reference to this, "at the prices which prevailed in 1907, the value of the by-products wasted in beehive coke-ovens was a little over $55,000,000."

But the result achieved by the chemical engineer in the working of this by-product over represents more than merely saving certain by-products. The gas produced can be separated by the perfect control of the method into "poor gas" for fuel purposes and "rich gas" for illuminating purposes so that the highest economy or conservation of values is thereby attained.

Another industry in which the chemical engineer has worked first for the purpose of exploitation and later for conservation of material and values is the starch industry. Starch, as we all know, is one of the most widely distributed vegetable products and has always played a great part in the world's supply of food. In Europe, it is potato and wheat starch, in the United States it is corn starch, and in tropical countries it is rice, tapioca, sago, etc., that are the important varieties of this cereal food. Not only has the production of starch been developed, however, for food purposes, but enormous quantities of the starchy substances serve as the starting point in the fermentation industries. Then we have the use of both starch and its alteration product dextrine in the textile industries and the production of glucose as the product of the hydrolysis of starch, and the manufacture of nitro-starch and its utilization in the explosive industry. All of these industries have attained a high degree of perfection by the application to them of a chemical understanding of the nature of starch and its alteration products and the devising of processes by which the several reaction changes could be carried out with exactness and economy. What that means as applied to one single branch of the starch industry, those of us who had the opportunity on the occasion of our meeting last June to go through the works of the Corn Products Company at Edgewater, N. J., can appreciate. The glucose production of the United States in 1907 is stated to have been 800,000 tons, requiring 40,000,000 bushels of corn as raw material. But besides the production of the solid grape-sugar and the liquid glucose for a great variety of uses, the separation of the germ of the corn from the starchy portion has made possible the production on a large scale of corn-oil, a product adapted for a wide range of uses—from soap making to the manufacture of rubber substitute.

Turning now to Inorganic Chemistry for an illustration, we have a splendid example of the work of the chemical engineer in the direction of conservation in the case of the natural and artificial nitrate industry, to which latter the Germans have already given the expressive name of "air-salpeter." The great source of nitrate for forty or more years past has been the deposits on the west coast of South America furnishing the so-called Chili salpeter or sodium nitrate. This has been drawn upon increasingly until in 1908 the quantity shipped was 1,730,000 tons, valued at $87,500,000. But the Chilean deposits are far from being inexhaustible. It is estimated that if the annual consumption increases merely by 50,000 tons, and this is to be reasonably expected, from 30 to 40 years will see the practical exhaustion of this supply. So in 1899, Sir Wm. Crookes startled the industrial world by calling attention, in what the newspaper men would call a "scare-article," to the nitrogen problem and the necessity of maintaining a supply of nitrogenous plant food if the world's food requirements were to be met. Crookes pointed out the way of relief when he said "the fixation of atmospheric nitrogen is one of the greatest discoveries awaiting the ingenuity of chemists. It is certainly deeply important in its practical bearings on the future welfare and happiness of the civilized races of mankind." The chemical engineer has responded nobly to this demand for conservation of available nitrogenous material by the working out of practical methods for the manufacture of what we called "air-salpeter." It must not be supposed, however, that success was easily obtained.

Not only were the usual experimental difficulties to be overcome but, as Prof. Bernthsen has well shown in his address before the London International Congress of Applied Chemistry in May last, there are theoretical difficulties of the most serious kind standing in the road of ready fixation of atmospheric nitrogen and oxygen in the form of nitrogen oxides convertible into nitrates. So it happened that the first large enterprise of this kind, the process of the Atmospheric Products Company, established at Niagara Falls, had to be given up as commercially unavailable. This was followed by the Birkeland and Eyde process, started in Norway with the cheapest water power to be found, and this is still in successful operation. A few years later (in 1905) the Schönherr process was worked out by the Badische Anilin and Soda Fabrik, also using Norwegian water power and still later the Pauling process at Gelsenkirchen, near Innsbruck, in Tyrol. The Schönherr process seems to be the most successful.

It produces a 40 per cent nitric acid, calcium nitrate, or calcium or sodium nitrite, according as the absorption part of the process is modified. All of the processes mentioned require the cheapest electrical energy which can only be developed by cheap water power, and thus far best developed in Norway. Prof. Bernthsen states, however, that probably within a few years the annual output of calcium nitrate or "air-salpeter" will reach 100,000 tons. As this involves first the fixation of atmospheric nitrogen, and second the use of "white coal," as water power is sometimes fancifully called, it is a true lesson in conservation of natural resources, especially as it also enables us to replace a rapidly disappearing natural product.

Closely related to this recently developed inorganic industry of air-salpeter is the slightly older one of calcium carbide and

its offshoot the cyanamide or "nitrolime" industry. With the production of calcium carbide in the electric furnace in 1892 by Willson and the publication of Moissan's work on the electric furnace in 1894, sprang into existence a great industry, as the acetylene gas lighting made possible thereby had great advantages. Isolated lighting plants, acetylene lamps for automobiles and carriages, luminous buoys and signals, a new material for lampblack manufacture and other utilizations all were rapidly developed. The perfecting of the furnaces and the process for this manufacture of carbide enlisted the attention of electrical and chemical engineers and at the present time the world's production of calcium carbide is estimated to be about 200,000 metric tons per annum, largely in countries like Norway, Italy and Switzerland where cheap water power is available, as well as in the United States at Niagara Falls and Sault St. Marie. This production, however, for the time being outran the demand for carbide for acetylene lighting.

Relief from over-production by finding new outlets and utilizations is always to be preferred to closing of works already in operation, so chemical engineers have found new possibilities for calcium carbide. By far the most important of these is the production from calcium carbide of calcium cyanamide by the action of nitrogen gas, as worked out by Drs. Frank and Caro. We have here an exothermic reaction in which nitrogen is absorbed by the carbide with the production of calcium cyanamide and carbon. This takes place at a temperature of from 800° to 1000° C., much below that needed for the carbide manufacture. Not only can all the nitrogen of this cyanamide be converted into ammonia by decomposition with steam, but it is gradually decomposed by the chemical and bacteriological constituents of the soil into ammonia, which becomes fixed by the vegetable mould and is so held. The cyanamide is also convertible into calcium cyanide by melting with fluxes, into dicyandiamide for dye-color manufacture, into urea, guanidine and other hitherto relatively expensive organic compounds. The dicyandiamide is already used as a "deterrent" in smokeless powder manufacture, reducing the temperature of the explosion without diminishing explosive force, and the crude calcium cyanamide with certain fluxes under the name of "ferrodur" is employed for case-hardening of iron and steel.

The statement is made that the works for the manufacture of "nitrolime" now in operation or in course of construction have a capacity of 166,000 tons per year.

The illustrations of conservation, whether from the point of view of better utilization of materials, or the production of new and varied products or the recovery of what were waste products, could be greatly extended did time allow.

We might refer to the way in which sulphur recovery has been worked out in the alkali industry, or to the manganese recovery in connection with the chlorine production from manganese dioxide, so that "recovered manganese" is to-day a most valuable article of commerce. Or we might note the saving resulting from the manufacture of reclaimed rubber, but these and similar illustrations of the conservation theme as influenced by the work of the chemical engineer will have to be passed by for the present.

That there are numerous problems as yet unsolved of equal and possibly greater importance than some of these discussed will also be conceded by those possessing even a moderate acquaintance with chemical industries.

One of these problems, for instance, is the recovery of the valuable constituents of the waste liquor of the sulphite wood-pulp process. A German authority states that every liter of this waste liquor contains 120 gms. of organic material as against 10 to 15 gms. of mineral substance. Dr. A. Frank estimated that in 1904 there was wasted in this way in Germany 300 million kilograms of organic material, concerning which we know that it has value in several directions. In this country, the sulphite wood pulp process has also an extensive development and the same waste liquor is run off into our streams.

A somewhat different problem, but one of even greater importance, is the loss of valuable metals in smelter smoke and fumes. The American production of bismuth is not over 10,000 lbs. a year and considerable amounts of bismuth and bismuth compounds are imported every year. Yet it is estimated in the forthcoming report of the U. S. Geological Survey for 1908 that 880 lbs. of bismuth per day are being thrown off in the smoke of the great Washoe smelter at Anaconda, Mont., and with this also go copper, lead, zinc, arsenic and other mineral products. One way of saving much of this lost material is pointed out in noting that the replacing of smelting methods by electrolytic methods, possible in some cases, allows these metals to be recovered from the deposited slimes.

In conclusion let us emphasize the fact that our chemical industries need not merely development or exploitation, but if they are to continue to flourish as we are drawn more and more into international competition they must have the newest and best results of chemical research applied by the experienced chemical engineer. New and better materials must be sought, better processes evolved, economies effected at all possible stages, and waste products carefully looked after. The most successful chemical industries in the world, which show the highest scientific and technical development and which are steadily expanding and throwing out branches, are those of the great German chemical companies. Organizations like the Badische Anilin and Soda Fabrik, which have by their employment of both research chemists and chemical engineers effected industrial revolutions, one after the other, like the introduction of artificial indigo, the contact sulphuric-acid manufacture, and last of all the production of air-salpeter or artificial nitric acid and nitrates, are those which reap the rich commercial rewards and for the reason that they have done the work and earned them. The hope of a successful American Chemical Industry lies in the same direction.

The Electric Shaft Furnace at Domnarfvet, Sweden.

A full and interesting account of the experiments made with the electric shaft furnace at Domnarfvet, Sweden (see our Vol. VII, p. 419), and the conclusions drawn therefrom has been contributed by Mr. LARS YNGSTRÖM to the *Jern-Kontorets Annaler*, No. 9, 1909, from which the following abstract has been made.

The author first makes a heat analysis of the smelting process and determines the amount of coal and electric energy necessary to make a given grade of pig iron, which is assumed to have the following composition: Three per cent C, 1 Si, 96 Fe, and traces of Mn, P, S, Cu.

Further, it is assumed that the ore consists of Fe_3O_4 with 60 per cent iron plus burnt lime, and that the escaping gases contain 30 per cent CO_2 and have a temperature of 200° C.

In making the computations, the following heat values are assumed:

To reduce 1 kg Fe from Fe_3O_4 requires 1650 cal.
" 1 " Fe " Fe_2O_3 " 1800 "
" 1 " Si " SiO_2 " 7830 "

To oxidize 1 kg C to CO_2 generates 8080 cal.
" 1 " C to CO " 2470 "

1 kw-hour corresponds to 857 cal.

The smelting and overheating of 1 kg of pig iron requires 280 cal.

1 kg. of slag[1] requires 595 cal.

Specific heat for dry furnace gas, 0.245 cal. per kilograms.

It is also assumed that all raw material introduced into the furnace has an initial temperature of 0° C., that the ore is free of water and CO_2, and that the lime is burnt.

[1] This figure may appear somewhat high, but it was thought advisable to increase it by about 100 cal. over the value obtained by experiment in the laboratory; partly on account of the high temperature of the slag and partly on account of the formation of carbide in the slag when electricity is used.

Amount of C required per 1000 kg of pig iron.

Reaction formulas:

(a) $13\,Fe_3O_4 + 40\,C = 12\,CO_2 + 28\,CO + 39\,Fe$.

(b) $13\,SiO_2 + 20\,C = 6\,CO_2 + 14\,CO + 13\,Si$.

According to formula (a) 960 kg of Fe are obtained from 1325.71 kg Fe_3O_4, and requires for reduction 210.99 kg. C, forming 576.70 kg of gas ($CO_2 + CO$).

According to formula (b) 10 kg Si is obtained from 21.43 kg SiO_2, and requires for reduction 6.59 kg C., forming 18.02 kg of gas ($CO_2 + CO$).

The total amount of carbon required is:

For the reduction of 960 kg Fe	210.99 kg
For the reduction of 10 kg Si	6.59 "
For carbon in the pig iron	30.00 "
Total chemically pure carbon	247.58 kg

Amount of Slag.

With a mixture of ore and lime containing 60 per cent Fe there is required 1600 kg of this mixture for 960 kg Fe. This corresponds to 1325.71 kg Fe_3O_4.

Corresponding amount of slag	274.29 kg
SiO_2	21.43 "
Amount of slag per ton of pig iron	252.86 kg

Amount of Gas.

According to form. (a) there is developed $CO_2 + CO$	576.70 kg
According to form. (b) there is developed $CO_2 + CO$	18.02 kg
Total per ton of pig iron	594.72 kg

Calorific Value of Fuel.

When 1 kg C combines with O, forming a gas which contains 30 per cent CO_2 by volume, there is generated $0.3 \times 8080 + 0.7 \times 2470 = 4153$ kg-cal.

Heat Balances.

Heat required:

Reduction of 960 kg Fe from Fe_3O_4 at 1650 cal.	1,584,000 cal.
Reduction of 10 kg Si from SiO_2 at 7830 cal.	78,300 "
Melting and overheating of 1000 kg of pig iron at 280 cal.	280,000 "
Melting and overheating of 252.86 kg of slag at 595 cal.	150,452 "
Heating of 594.72 kg $CO_2 + CO$ to 200° C. at 200×0.245 cal.	29,142 "
Total	2,121,894 cal.

Heat developed:

Combustion of 217.58 kg C at 4153 cal.	903,610 cal.
Heat developed by electric current	1,218,284 "
Total	2,121,894 cal.

Therefore, the theoretical amount of electric energy required is

$$\frac{1,218,284}{857} = 1420 \text{ kw-hours.}$$

The curves in Figs. 1 and 2 are based upon calculations similar to the foregoing. The amount of carbon in coke and charcoal being assumed as follows:

Coke	C	= 85 per cent
	H_2O	= 5 " "
	Ash	= 10 " "
Charcoal	C	= 85 per cent
	H_2O	= 12 " "
	Ash	= 3 " "

Fig. 1 applies to iron reduced from Fe_3O_4, while Fig. 2 applies to that reduced from Fe_2O_3.

The experiments at Domnarfvet began in the spring of 1907, and with but few interruptions have continued up to the present time. The electrical equipment installed especially for these tests consists of a 7000-volt, 60-cycle, 900-hp synchronous motor direct connected to a three-phase, 25-cycle generator which supplies energy to a transformer set up near the furnace. The generator permits of an adjustment of e.m.f. in small steps from 300 to 1200 volts. This wide range of voltage has made it possible to test furnaces under various conditions and of various designs with the most favorable voltage. The fields of the synchronous motor and generator are excited by a 20-kw, 220-volt, direct-current generator mounted on the same shaft. The set is started by a 500-volt, three-phase induction motor also mounted on the same shaft. The machines are all equipped with instruments which permit the calculation of the energy and power being used.

From the switchboard the electric energy is carried by copper bars laid in conduits under the floor direct to the three-phase

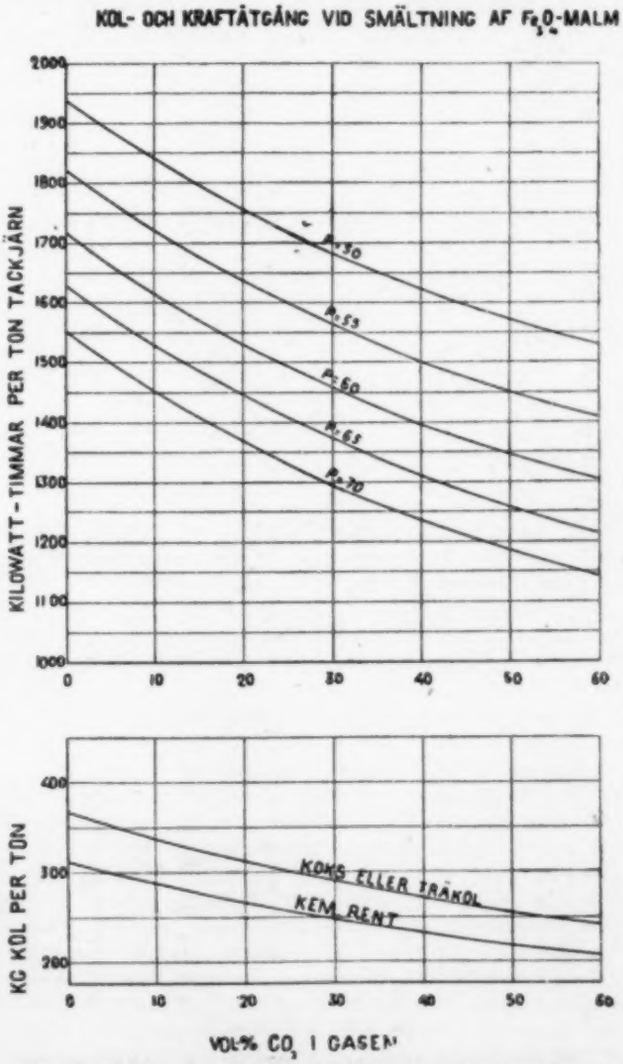


FIG. 1.—ENERGY AND CARBON REQUIRED FOR SMELTING IRON FROM Fe_2O_3.

(The abscissas in both diagrams are per cents. by volume of CO_2. The ordinates of the upper diagram are kilowatt hours per ton of pig iron. The five curves refer to 50, 55, 60, 65, 70% Fe in the ore. The ordinates in the lower diagram are kg per ton. The upper curve refers to coke or charcoal, the lower to chemically pure carbon.)

air-blast transformers. These transformers have a combined capacity of 1500 kilovolt-amperes with a transformation ratio of 14 to 1—that is, by means of voltage control at the generator, the secondary voltage of the transformer can be changed in small steps from 20 volts to 80 volts.

For controlling the operation of the furnace an instrument board is mounted nearby which contains one three-phase precision wattmeter for unbalanced phases; three ammeters, one for each phase, and one voltmeter with switch for connecting to various phases.

The ammeter and wattmeter are connected through series transformers to the leads. The voltmeter and wattmeter potential coil are direct connected to the leads. Right under the instruments are placed the hand wheels for controlling the position of the electrodes in the furnace.

The first experiments were performed with an electric furnace constructed in accordance with certain patents owned by Messrs. Grönwall, Lindblad and Stalhane, who together formed a company Elektrometall in Ludvika. These gentlemen, together with the Grangesbergs Company, were also interested in the continued experiments.

The construction of the first furnace is shown in Fig. 3. It

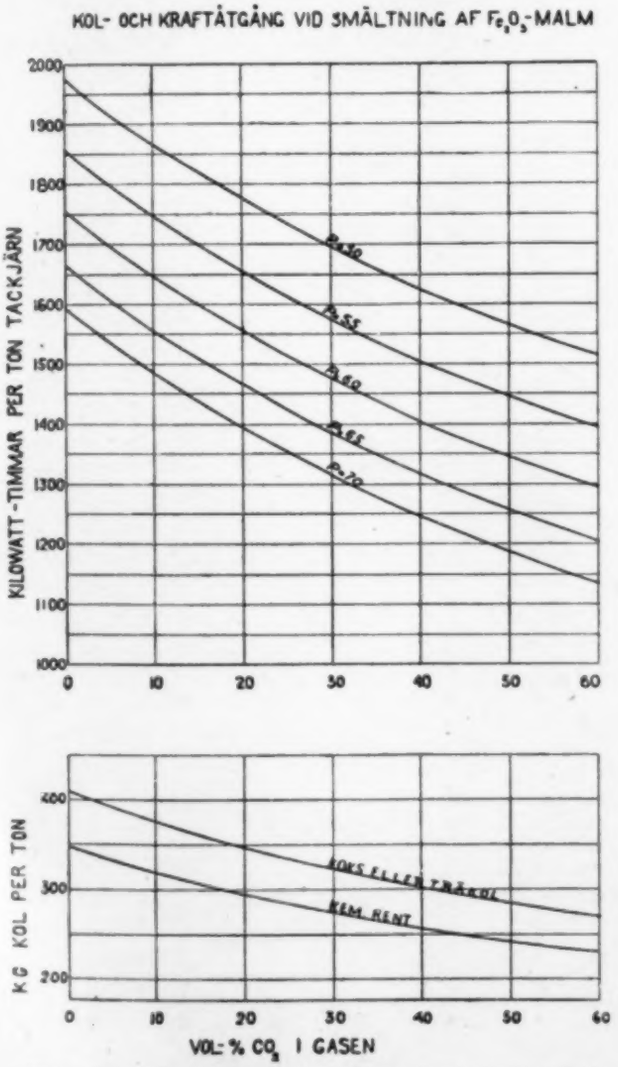


FIG. 2.—ENERGY AND CARBON REQUIRED FOR SMELTING IRON FROM Fe_2O_3.

(The abscissas in both diagrams are per cents. by volume of CO_2. The ordinates of the upper diagram are kilowatt hours per ton of pig iron. The five curves refer to 50, 55, 60, 65, 70% Fe in the ore. The ordinates in the lower diagram are kg per ton. The upper curve refers to coke or charcoal, the lower to chemically pure carbon.)

is a shaft furnace with the hearth of tamped quartz. In the bottom of the hearth there are large grooves. The one in the middle leads to the tap hole, while the other two on the sides lead to wells which are filled with iron. The bottoms of these wells are made of copper plates covered with pressed graphite blocks, each of these forming the electrodes of the furnace.

The furnace is started with blast as in the ordinary shaft furnace, and when a sufficient amount of iron has collected in the hearth, the blast is turned off and the electric energy turned on. It was intended that the electric current entering the furnace through the iron on the one side, passing through the furnace on the other side would, partly by the overheating of the molten iron and partly by the energy lost by the resistance in the path, liberate enough heat to melt the ore.

For short periods of time this furnace could operate, but it was found impossible to obtain a hearth bottom which could stand the high temperatures. Therefore, this furnace was rebuilt in the form shown in Fig. 4. The same principle is used

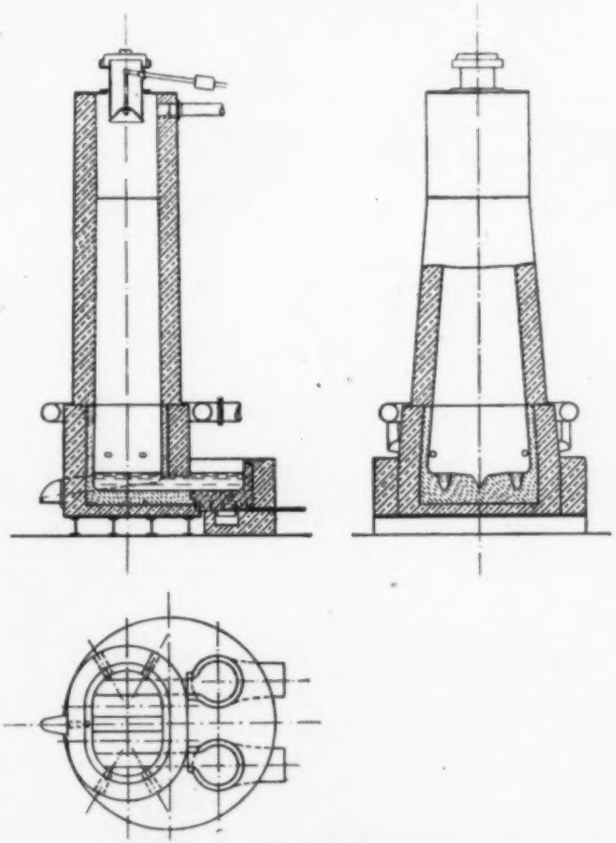


FIG. 3.—FIRST DESIGN OF ELECTRIC SHAFT FURNACE. (TWO BOTTOM ELECTRODES.)

as in the first furnace, but the lower part is more solidly constructed. The electric energy enters and leaves at opposite sides of the furnace. The most important changes consisted in using magnesite. This material gave very much better results than were obtained in the first furnace. However, it had the disadvantage of being a relatively good conductor of electricity at very high temperature.

These first experiments clearly indicated that the solution of the problem could not be accomplished with furnaces where the current was introduced into the molten iron at the bottom of the hearth. Therefore, the next furnace was built along different lines.

The first one of this type, shown in Fig. 5, was a small shaft furnace with one graphite electrode in the bottom, and two diagonally opposite electrodes in the walls of the furnace at a certain height above the bottom. During the experiments the upper electrodes became considerably burnt and had to be exchanged for ordinary adjustable carbon electrodes fitted with water-cooled jackets. The connections were so made that the current could be conducted either across the shaft from one electrode to the opposite one, or through the bottom electrode. This furnace proved much more durable than the former ones, but had the disadvantage that the walls of the shaft were badly damaged around the electrodes, due to the very large amounts of energy there liberated.

If it could be so arranged that the electrodes were surrounded by ore, the heat which in this furnace was lost in heating up the walls, could be turned to good account, and the walls made

to last a great deal longer. In accordance with these ideas a furnace was constructed as shown in Fig. 6, and experiments with this one showed that the problem was very near its successful solution and led to the construction of the large shaft furnace which forms the subject of this article. The construction of this furnace is shown in Fig. 7, which in appearance is very similar to an ordinary blast furnace in which the tuyeres

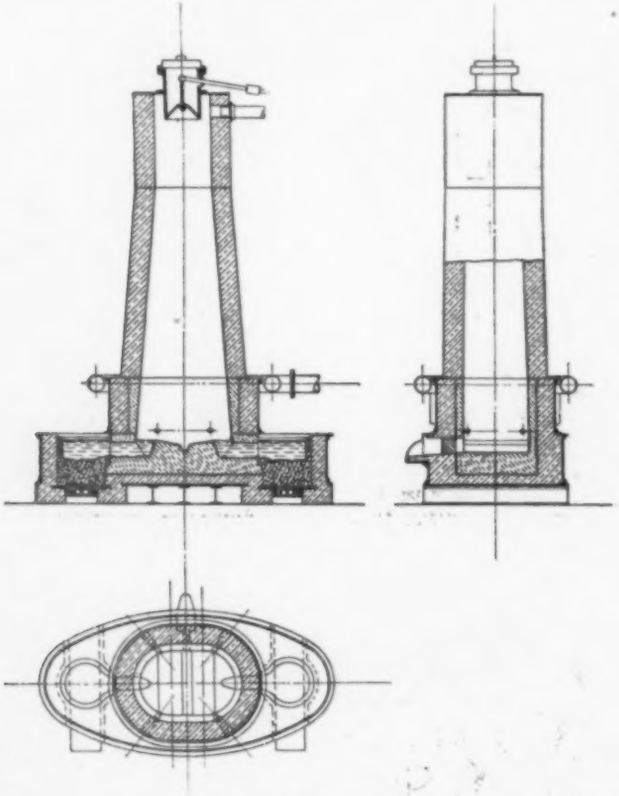


FIG. 4.—SECOND DESIGN OF ELECTRIC SHAFT FURNACE. (TWO BOTTOM ELECTRODES.)

are replaced by electrodes. This furnace was described by Dr. Eugene Haanel in a paper before the Electrochemical Society, May 6, 1909 (this journal, Vol. VII, p. 251, June, 1909).

The melting chamber is in the form of a crucible about 2250 mm in diameter and 1500 mm high and lined with magnesite. As the charge descends through the opening in the top of the melting chamber, free space is left between the charge and the upper walls. This point is of great importance and has much to do with the success of this type of furnace. Experiments have shown that the angle assumed by the descending charge is about 50° or 55° with the vertical.

This free space around the roof of the chamber is utilized for cooling the walls and electrodes by drawing comparatively cool gas from the tunnel head and forcing it through tuyeres into the space at the top of the crucible, as shown in the illustration. This method of cooling is extremely efficient, since the heat absorbed by the gases is given off to the charge as the gases ascend to the top of the shaft, and, therefore, no heat is lost except by radiation from the tuyeres. By this method of circulation of gas a better distribution of heat throughout the charge is obtained than is usual with other types of electric furnaces.

The electrodes are introduced through the roof of the melting chamber passing through the free space into the charge, which is made up of two square carbons 33 cm on a side, having a total cross-section of 2180 sq. cm. The electrode holder consists of a strong steel frame which supports wedges by means of which the copper plates conducting the electricity from the cables are pressed against the faces of the electrode. The position of the electrodes is controlled by the steel cable which passes by means of pulleys to a winch situated at the instrument board opposite the furnace. The external part of the electrodes is protected with an armored asbestos covering to prevent oxidation in the air, and the stuffing boxes are water-cooled, special devices, not shown in the illustration, being provided to prevent the gas within the melting chamber from leaking out around the electrodes.

The shaft, which is 5.2 meters high and 1.525 meters internal diameter at the largest section, rests upon an iron framework supported by six cast-iron columns. This construction permits repairs to be made on the melting chamber without in any way disturbing the shaft itself.

This furnace was put into operation the seventh of last May, and with the exception of a few slight interruptions was run continuously to last July, and could have continued for an indefinite period had it not been necessary for other reasons to shut it down.

The furnace is started as an ordinary blast furnace with the electrodes removed, then several charges of charcoal and coke are put in, the electrodes are inserted and the shaft filled with ore and lime, together with charcoal and coke. The amount of charcoal used is reduced continually and at the end of the first week the charges consist of 100 kg ore, 3 kg lime (burnt),

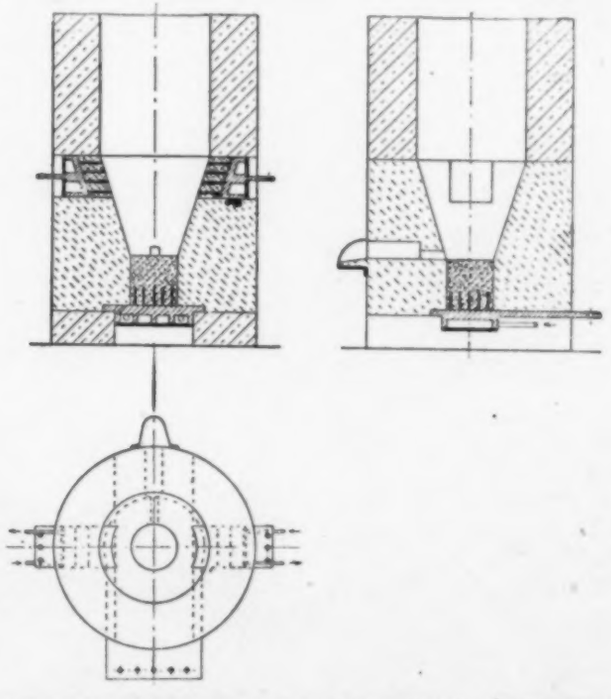


FIG. 5.—THIRD DESIGN OF ELECTRIC SHAFT FURNACE. (ONE BOTTOM ELECTRODE AND TWO ELECTRODES IN WALLS.)

24 kg coke, and from that time on 100 kg of ore, 4 kg of lime and from 22 to 24 kg coke. The greatest number of charges per day when operating with coke was 62. The shaft held 72 charges, and, therefore, the number of times it was filled per day was 0.86.

When operating with charcoal the number of charges per day was 80, and since the shaft then held only 41, the number of times the shaft was filled per day was 1.95. When using charcoal alone the amount of carbon per 100 kg of ore varied from 21 to 28 kg. The charcoal was measured by volume, 1 hectoliter being taken as 16 kg, because of its relatively high moisture content.

At the beginning, when the furnace was still cold in the bottom, the current existed principally under the arch. The path of the current followed close under the arch, and, therefore, the latter was considerably heated and its surface partially melted. As soon as the gas circulation was put into operation

its cooling effect was very effective, the temperature of the arch sunk and the heat descended toward the bottom of the crucible. As the lower part became heated, the resistance was lowered, and, therefore, the current gradually worked down toward the bottom of the furnace. After the furnace was in normal operation, the gas circulation could be discontinued for several days without any tendency for the current to rise toward the roof of the crucible. Charcoal in the charge seems to assist in holding the heat near the bottom.

In several cases it happened that the quantities of iron produced did not correspond to the percentage in the ore. For example, in one week from 48 tons of ore only 17 tons of iron, which corresponds to 27 tons of ore, was obtained; while in the following week from 22.6 tons of ore, 23 tons of iron were obtained. This can be explained only by the fact that more or less of the reducing iron clung to the walls and bottom of the furnace in the first case, and was drawn off in the second.

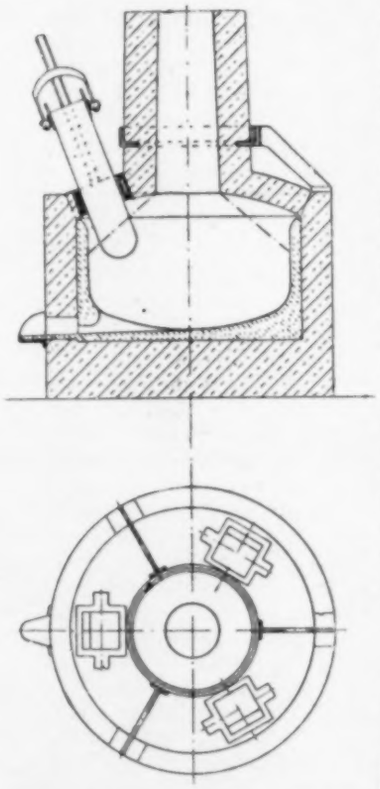


FIG. 6.—ELECTRIC SHAFT FURNACE. (THREE ELECTRODES IN WALLS.)

The furnace was sometimes cold on account of there being too much carbon in the charge. However, this was quickly overcome by increasing the amount of ore. Moreover, such an occurrence can be easily prevented by proper construction of the furnace walls. The slag indicates plainly an excess of carbon in the charge, in which case calcium carbide is formed and an odor of acetylene gas is noticeable.

In certain cases the furnace showed a tendency to hang, but after the amount of charge along the walls had reached 80 per cent of the total this difficulty disappeared. Experiments proved that an electric furnace can be charged with ore dust if all the dust is introduced along the walls and the height of the charging head is so chosen that the upper surface comes somewhat under the gas-collecting cylinders so that the dust cannot be drawn over into the gas tuyeres. The briquetting of ore dust is, however, not necessary with electric furnaces.

A few slight mishaps during the operation of the furnace had a certain influence on the results obtained. For example, near the end of the first week the water jacket of one of the electrodes sprang a leak so that it was necessary to operate the furnace for a time with only two electrodes. Again, on account of failure of one of the electrical machines, the transformer at two different times was connected directly to the high-tension line and operated at 60 cycles instead of 25.

While in operation no surges of current or sudden peaks were noted. Even when tapping the furnace the measuring instruments indicated no fluctuation, and, therefore, the electrodes required very little attention, being regulated about once a day, and in one case they ran five consecutive days without any adjustment. The largest current carried by the electrodes was about 9000 amp per phase. The power factor at 25 cycles was from 0.8 and 0.9, while at 60 cycles it was about 0.7.

The amount of energy which can be delivered to the furnace depends largely upon the internal resistance of the mass; the larger the resistance, the greater the amount of energy which can be developed without increasing the current to excessive values. With 25 cycles, 80 volts was sufficient to produce the proper current, and with 60 cycles, 65 volts was necessary. The voltage impressed upon the furnace depends principally upon the fuel used and its relation to the ore. Table 1 shows this relation.

TABLE 1.

Charge—	Volts across electrodes	Current per electrode in amperes
Coke in excess	34	9,600
" not in excess	36	8,800
Charcoal in deficiency	60	6,300
" right	54	7,600
" in excess	48	7,600
Coke and charcoal in excess	35	9,200
" " " right	48	7,600

In operation the furnace proved to be very steady and easy to care for, and it was tapped about every six hours.

Seven different kinds of charges were used in the tests. Since the principal purpose of the experiments was to develop a practical furnace construction, the charge at first consisted of a rich, easily reduced hematite ore which had a considerable phosphorus content. As a result, however, later on it was impossible to produce a first-class pig iron from first-class ore, because the iron was always adulterated by phosphorus from the ore which remained on the walls of the furnace.

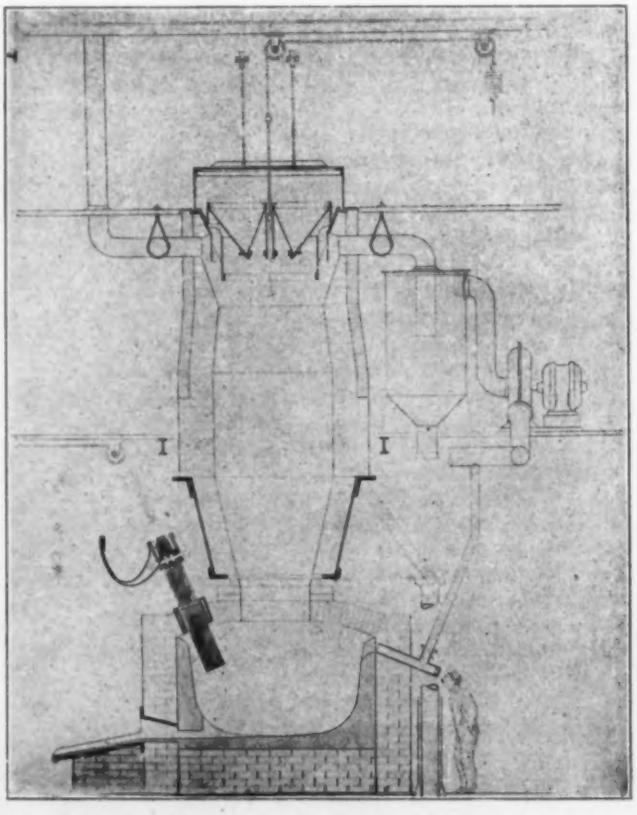


FIG. 7.—FINAL DESIGN OF ELECTRIC SHAFT FURNACE.

Burnt Lime.		Coke.		Charcoal.	
Fe_2O_3	0.5%	C	85%	C	65-80%
MgO	2.0%	H_2O	4%	H_2O	12-20%
CaO	96.5%	O + N	1%	Ash	3%
Al_2O_3	0.2%	Ash	10%		
SiO_2	0.2%	(0.5-0.7% S)			

The composition of charge mixtures was as follows:

Mixture No. 1.

826 charges of
100 kg Grängesbergs Skarnings ore.
4-5 kg lime (burnt).
10-26 kg coke.
10-0 charcoal.

Mixture No. 2.

575 charges of
10 kg of Langgrufve ore (briquettes).
11-18 kg of lime (burnt).
0-21 kg of coke.
20-0 kg of charcoal.

Table 2.—Ore Analyses.

Ores from—	Fe_3O_4	Fe_2O_3	FeO	Mno	Mgo	CaO	Al_2O_3	TiO_2	SiO_2	P_2O_5	H_2O +CO_2	Fe	P	S	Mixture No.
	%														
Grängesberg:															
Skärnings ore	13.75	71.65	—	0.12	0.74	6.80	0.35	—	3.02	4.19	—	60.11	1.83	trace	1 and 3
Langgrufvan:															
Briquettes	5.89	76.36	—	0.24	2.66	3.28	—	—	11.52	0.009	—	57.70	0.004	trace	2
Magnetically separated dust	80.89	—	—	0.13	2.80	2.96	2.14	—	11.30	0.009	—	58.57	0.004	trace	6
Johannisberg	58.58	—	6.78	5.58	5.61	4.98	0.91	—	5.56	0.007	11.72	47.69	0.003	0.13	6
Skottgrufvan:															
Vask ore	66.37	—	2.16	0.33	3.19	12.18	2.46	—	4.50	0.037	8.44	49.73	0.016	0.00	6
Tuollavaara	92.46	1.99	—	trace	0.03	1.28	trace	0.90	1.80	0.007	—	68.31	0.003	0.09	4 and 5
Taberg	74.48	—	—	0.42	4.96	4.90	—	—	15.30	0.014	—	53.93	0.006	0.00	5
Finnmossen	79.64	—	—	0.29	3.78	3.30	4.31	—	8.55	0.023	—	57.66	0.010	0.00	5
Nordmark	68.58	—	—	0.60	5.50	6.27	1.95	—	16.90	0.023	—	49.61	0.010	0.00	5
Brazil:															
Hematite	—	91.72	5.78	trace	0.25	0.20	trace	—	0.80	0.126	1.0	68.8	0.055	0.03	7

Mixture No. 3.

959 charges of
100 kg Grangesbergs Skarnings ore.
2-4 kg of lime (burnt).
8-13 kg of coke.
13-7 kg of charcoal.

Mixture No. 4.

599 charges of
100 kg Tuollavaara ore.
4 kg of lime (burnt).
7-11 kg of coke.
17-11 kg of charcoal.

Mixture No. 5.

623 charges of
20 kg Tuollavaara ore.
80 kg equal parts of Finnmosse-Tabergs and Nordmarks ore.
5 kg of lime (burnt).
4-0 kg of coke.
18-26 kg charcoal.

Mixture No. 6.

No. of Charges.	Langgrufve Ore.	Skottgrufve Ore.	Johannisberges-grufve Ore.	Lime.	Charcoal.
97 parts kg....	50	20	30	6	21
13 " "....	50	20	30	..	21
168 " "....	35	15	20	..	16
160 " "....	45	10	15	2	16
202 " "....	55	10	10	2	16
81 " "....	60	10	5	3	16
96 " "....	75	..	..	4	16

Mixture No. 7.

362 charges of
64-50 kg Brazilian ore.
2-0 kg Lime (burnt).
10 kg charcoal.

Since no air can enter the furnace, providing it is well built and properly run, the amount of fuel for different ores depends entirely upon the amount of carbon in the fuel and also upon the amount of this carbon which is oxidized to CO_2. In short, the amount of carbon used under normal conditions is always the same as would be theoretically required. The results of experiments demonstrate the truth of this statement. The carbon required can be expressed mathematically in terms of the CO_2 in the furnace gases. If M denotes volume per cent CO_2 in the gases, and k the percentage of carbon in iron, the carbon required in kilograms per metric ton of pig iron, if iron occurs in Fe_3O_4, is:

$$C = \frac{321.43\,(100 - k)}{100 + M} + 10\,k.$$

If Fe occurs in Fe_3O_4, we have

$$C = \frac{285.7\,(100 - k)}{100 + M} + 10\,k.$$

These formulas hold for chemically pure carbon and gas which is made up exclusively of CO_2 and CO.

It is evident that the circulation of the furnace gases has a definite influence on the amount of CO_2 in the gases. The experiments showed that the CO_2 content increases with the gas circulation, and that the temperature of the gases is very considerably increased. For instance, when operating with mixture No. 3, the CO_2 content of the gas was increased from 14.5 with no gas circulation to 39 per cent with constant circulation, and the corresponding temperatures were 75 deg. C. in the first case and 220 in the second. The experiments also indicated the existence of a certain circulation velocity which was best in each particular case, and that an increase over this velocity increased the temperature of the gases accompanied by a decrease in CO_2 content which was quickly noticeable.

What velocity to use in each case must be determined by experiment, and will depend to a certain extent upon the volume of the shaft. No exact measurement was made of the flow of gas used in these tests, but it may be estimated as about four or five times the rate at which the gases were generated in the furnace.

The high hydrogen content of the gas is noteworthy. This is partially due to the hydrogen in the charcoal, but to a greater extent to the moisture in the fuel, because when the moist gases are blown into the lower part of the furnace where the temperature is very high, hydrogen is developed.

A question which assumes great importance when judging the economical results which can be obtained with electric furnaces is that of the life of electrodes. With mixtures Nos. 1, 2 and 7 the consumption of the electrodes, on account of certain disadvantageous conditions, was abnormal; therefore, in the results given in Table 3 these tests have been omitted.

The electrodes were of Swedish make, and consisted of retort carbon with a permissible current density of 4 amp per sq. cm. The experiments indicate that the figures for the burning of the carbon can be reduced by about 40 per cent by using a higher quality of electrode.

Still more certain is the fact that the electrode consumption can be greatly reduced by suitable design. For instance, the electrodes used in the experiments were only about half consumed when they had to be discarded on account of their insufficient length. Therefore, if some construction be devised which would permit the continuous feeding of the electrode, a great saving would be accomplished.

Of great importance is the electrical efficiency of the furnace. By electrical efficiency is understood the ratio between the electrical energy used and the amount of energy required to reduce the ore. The electric current in the furnace produces the following effects: (1) It raises the mass to the proper temperature. (2) Melting and overheating of pig iron and slab. (3) Driving off of H_2O and CO_2. (4) Heating of the gases. The sum of these quantities of energy, less the heat which is developed by reduction of carbon by oxygen in the ore, to a gas of greater or less CO_2 content, is just the amount of energy which theoretically must be supplied from the electric circuit, and which is denoted as the theoretical electric energy. The losses of electric energy are as follows: (1) Heat in the cooling water around the electrodes. (2) Heat in leads and contacts. (3) Radiation from the walls of the furnace.

In tabulating the results obtained with the various mixtures, Nos. 1, 2 and 7 have been omitted, because during the first two runs the furnace was not yet in normal operating condition, while the last run was too short to give consistent and reliable results.

TABLE 3.

Mixture No.	Electrode burning. Kg. per ton	Electrode consumption. (metric) pig iron.
3	1.4	11.1
4	7.0	12.7
5	8.1	14.3
6	6.9	19.4
3—6	5.2	13.8

The losses varied between 230 and 270 kw, and were higher near the end of the experiments than in the beginning. These losses may be divided approximately as follows:

Cooling water	118 to 125 kw
Loads in contacts	40 kw
Radiation	110 to 180 kw
Total losses	239 to 275 kw

By building furnaces of higher capacity, for instance of 2500 hp, and by paying particular attention to the factors which have a bearing on the heat losses, it is not impossible that a furnace can be built with an electric efficiency of about 80 per cent—that is, one which surely could produce more than three metric tons of pig iron every per horsepower year, provided a rather rich ore is used.

It is difficult to estimate from the experiments which have been made what the maintenance cost of a furnace would be. The present furnace, however, has been in continuous operation for 85 days, and successfully met all requirements which indicate a durable furnace.

The furnace's weakest point is the arched roof of the crucible, but by cooling with circulating gas this can be prevented from burning out. The bottom of the crucible and the walls of the shaft did not show any appreciable ill effects from the heat more than the ordinary shaft furnace. It has happened that the masonry back of the electrodes became red hot, but by suitable cooling devices, utilizing circulating gas, this can easily be overcome. Moreover, if the masonry should burn out at these points, it is easy to repair them without interfering with the operation of the furnace. Therefore, by suitable design and proper material in the walls, the durability of the furnace can be rendered thoroughly satisfactory.

With electric furnaces it appears that the process of producing iron of different compositions is under more exact control than in other processes, since it consists mostly in regulation of voltage, which will produce corresponding changes in temperature. It has been possible to produce iron with less than 1 per cent carbon, but in this case the slag content had a noticeable amount of iron. As a rule, the carbon content varied between 2 and 3 per cent.

With an electric furnace it is very easy to increase the silicon content in the pig iron. Analysis has shown 3.61 silicon, but it is also possible to reduce the silicon content as low as from 0.3 to 0.5 per cent.

It can be counted on that all the phosphorus in the charge will enter into the composition of the iron, but it should be noted here that in electric furnaces only one-third as much fuel is used as in the ordinary shaft furnace, and that in the production of first-class charcoal pig iron it is the charcoal which supplies the phosphorus. Therefore pig iron produced in an electric furnace will have a lower phosphorus content than that produced in an ordinary blast furnace.

A special advantage of the electric furnace is the possibility of driving off sulphur. Using a mixture of about 5 per cent sulphur, an iron with less than 0.005 per cent sulphur was produced. The requirements for driving off of sulphur are as in electric steel furnaces, a basic slag and a high temperature. The calcium carbide formed by electric current also plays an important part in the driving off of the sulphur.

Judging from the tests made at Domnarfvet, it appears that the production of iron from iron ore in electric furnaces is successfully accomplished, both technically and economically speaking, but in order to obtain a more complete knowledge of the subject, these experiments should be continued on a larger scale. If electric operation is to be worth while, however, sufficient electrical energy at low price must be available. Summing up the advantages of the electric shaft furnace over the ordinary blast furnace, they are as follows:

1. Lower installation cost, since blowing engines and heating apparatus are not required.
2. Saving of about two-thirds of the fuel required in an ordinary blast furnace.
3. Ore dust can be used without briquetting.
4. The generated gases have a high heat value when free from nitrogen.
5. The possibility of producing in the furnace itself low-carbon iron.
6. Less attendance, since the transportation and charging of two-thirds of the fuel used in ordinary shaft furnace is done away with.

Evidently it would be advantageous to combine electric shaft furnace for iron ore reduction with an electric steel furnace, which would refine the melted iron from the electric shaft furnace.

We are obliged to Mr. Yngström for a copy of his paper from which this translation was prepared, and to the editor of *Jern-Kontorets Annaler* for placing the cuts of the illustrations at our disposal.

TABLE 4.

Mixture No.	Carbon required (chem. pure).	Energy required calculated from amount of carbon, kw-hr per metric ton.	Actual energy used, kw-hr per metric ton.	Electric efficiency.
3	252	1470	3114	0.47
4	254	1438	2473	0.58
5	284	1741	3245	0.54
6	294	1870	3334	0.56

Rapid Method for the Determination of Chromium in Chrome and Chrome-Nickel Steel.

BY N. M. RANDALL.

Dissolve one gm. steel in 25 cc nitro-sulphuric acid. When in solution remove from the heat and add about 15 or 20 cc cold water. Rotate the liquid in the beaker and drop in at once about one gm. sodium bismuthate and continue to whirl for a few seconds.

Replace on the light and boil hard, the permanganate formed

by the sodium bismuthate is rapidly decomposed and there remains a clear violet liquid (manganic metaphosphate) which on further boiling converts the remainder of the chromium to CrO_3.

Decompose excess of manganic metaphosphate with ½ cc or more if necesary of dilute HCl; boil one minute, cool somewhat, dilute to 200 cc with cold water, add slight excess ferrous ammonium sulphate, and titrate with permanganate.

The end reaction is very sharp and leaves no doubt in the operator's mind.

The results are accurate and an analysis can be completed in 10 minutes.

This method has not yet been tested in presence of W and Mo.

Acid for dissolving steel:

300 cc nitric acid, 1.42 sp. gr.

300 cc sulphuric acid, 1 pt. acid, 3 pts water.

300 cc water.

100 cc phosphoric acid, 85 per cent.

1½ grams manganese sulphate.

Sparrows Point., Md.

Uniformity of Blast Furnace Operation.

By Jos. W. Richards.

One of the great essentials for the attainment of high efficiency of furnace operation is the careful striving of the engineer for uniformity of working conditions. In practice it is the irregularities which fetch the superintendent out of bed at midnight, keep him on duty 24 or 48 hours at a stretch, and unduly sprinkle his hair with silver. What engineer has not experienced the sense of restful satisfaction which comes from seeing a furnace at last operating regularly, and has not felt at the same moment the conviction that his entire future problem was the preserving of those conditions by uniformity in everything pertaining to the furnace and its operation?

The history of blast furnace practice, when intelligently understood and properly appreciated, is mostly a record of efforts to attain greater uniformity of operation. The directions which these efforts have taken are in regard to the quantity, dryness and temperature of the blast; the composition, size of lump and relative proportions of the fuel, flux and ore; the times of charging, and the distribution of these within the furnace; the running out of pig iron and slag; the quality and temperature of iron and slag; the preserving constant of the interior shape of the furnace. We will discuss each of these in detail.

Composition of ore, flux and fuel.—No furnace can run successfully or uniformly unless the materials charged are of fairly uniform composition, both individually as ore, flux and fuel, and collectively as charge. To this end managers prefer to have large sources of supply of uniform materials. An iron ore mine which yields millions of tons of ore a year of nearly constant quality is an asset whose value to the blast furnace plant cannot be too highly estimated. Our Lake Superior ores are pure and rich, the large Alabama deposits are impure and moderately rich, the "Minette" of Luxembourg is impure and lean, yet all these are highly valuable deposits because of their immensity and tolerable uniformity. To find more deposits of similar constancy, iron works seek as for a jewel of great price.

Similar considerations govern the supply of flux and fuel. Large quarries of limestone of uniform quality, an abundant supply of coke of constant composition, are worth the best efforts of the works to obtain, and give comfort to the management when procured.

Failing the above conditions, when several varieties of ores, fluxes or fuels in varying proportions have to be worked, the furnace depends for its proper and successful working upon the chemical laboratory. Each variety of ore must be analyzed and the several varieties then mixed in proper proportions to obtain a uniform ore mixture. It is only in exceptional cases that this is not an absolutely necessary preliminary, since large deposits of uniform material are few and far between.

Given satisfactory and uniform quality of materials brought to the plant, or proper ore mixture, flux mixture or mixed fuel at hand, they should be preserved from rain and snow in order that they enter the furnace substantially dry. The charging one day of dry materials and the next of wet materials soaked by heavy rain will necessarily cause irregularities in the furnace working; use the charges either dry, moist or wet, but whichever it may be, uniformly so, and another cause of irregularity is scotched.

Size of Lump.—All of us have wondered, on first looking at the charges being dropped into a blast furnace, how the furnace could digest them in such huge lumps. And the query is a reasonable one; the blast furnace has a foot, belly and throat, but it has no teeth or anything corresponding to masticating apparatus, and it only too frequently suffers from indigestion. We should help it by "Fletcherizing" the charges. The old-fashioned manager who says it does not make any difference, and throws in lumps of ore, flux and fuel as large as a football, simply does not know that uniformity of lump, in fairly moderate size, is a great advantage to the furnace. One furnace in the United States, upon the advent of a new manager with the above idea in his head, increased its output 25 per cent, accompanied by much greater regularity of working, by merely sizing the ore, flux and fuel down to that uniform lump size for each which by experimenting was found to give the best efficiency.

Composition of the Charge.—Given fixed or uniform qualities of ore, flux and fuel, the proportioning of these so as to have the proper relations between them is in the hands of the chemist. He knows the analysis of each constituent, and calculates the charge upon the two requirements of (1) a given composition of slag to be produced and)2) a given quantity of inert material or burden (ore plus flux), melted down per unit of fuel used. The most important factor in the regularity of furnace working is the one under consideration. If this is faultily or improperly attended to, nothing can save the furnace from working with continued irregularity.

In making these calculations the chemist relies primarily on the accuracy of his analyses and upon experience as to what is required. By experience we mean that he must know from numerous tabulations of analyses of proper slags, within what limits of composition he must bring his slag in order to have it work properly; from similar tabulations he must know the maximum quantity of ore plus flux which unit weight of his fuel can properly take care of in the furnace. He thus draws on his experience, or the recorded and reliable experience of others, as the working basis or assumptions on which to base his calculations. All this must be accurately and reliably worked out; if not, uniformity of furnace working is impossible from the start.

Intervals of Charging.—Three meals a day is a good régime for others, but the blast furnace would soon be "dead" on it. Most furnaces are charged at intervals of five minutes to half an hour; neither of these is the ideal of proper feeding. Uniform driving requires uniform charging. Intermittent charging causes a varying height of charge column, a varying back pressure of blast in the furnace, a varying temperature of the issuing gases, and discontinuously varying layers of material descending the furnace shaft. The ideal charging method is undoubtedly continuous charging, without interruption. This will insure continuity in place of the above-mentioned irregularities. It has not yet been adopted for any blast furnace, as far as the writer is aware, but the constant feed of mechanical stokers, and particularly of certain modern gas producers, has certainly laid the foundation for the solution of this problem. The ideally regular working furnace of the future must necessarily have constant and continuous charging. Modern furnaces are approximating this by reducing to the smallest possible time the intervals between charges.

Distribution of the Charges.—An enormous amount of thought and effort has been given to the subject of securing uniformity in this respect. The closed top and mechanical distrib-

utors have done much to overcome the difficulties and to substitute mechanical regularity for human fallibility and carelessness. Every blast furnace manager will admit that the devices so far employed are not yet perfect; there is large room for their improvement. Coke will gravitate towards the middle and ore towards the sides, or *vice versa*, with attendant irregularity in the descent of the charges and working of the furnace. The mechanical engineer can here greatly help the metallurgist to solve a difficult and an important problem affecting uniformity of furnace operation.

Running Out of Iron and Slag.—Present practice is essentially discontinuous in this respect, with consequent irregularity in the working of the furnace. Pig iron is tapped two, three or four times a day, slag is usually tapped every one or two hours. It was formerly the practice to shut off the blast while tapping, but the irregularity caused by this was so great that it has been successfully abandoned. A few furnaces run off slag continuously. This solves half the problem, or advances half-way towards ideal regularity.

The continuous running out of slag and matte is a *fait accompli* in copper and lead blast furnaces, and has really been a most important improvement over discontinuous tapping, with the consequent irregularity of working of the furnace. Attempts to apply these principles to the iron blast furnace have succeeded so far only with regard to the slag. It is highly desirable, from the standpoint of uniform working, that both iron and slag should flow continuously from the furnace; whether through siphon taps or large settlers, etc., is a matter of detail. If this were accomplished, the sudden rushes of hundreds of tons of molten material can be dispensed with, casting machines of limited size could take care of the pig iron, small ladles handle the slag, and, best of all, the conditions in the crucible of the furnace, as far as concerns the presence of melted slag and iron, be kept perfectly uniform.

Interior Shape of the Furnace.—In the "days of old" this was a cause of secular irregularity which often resulted in the "blowing out" of the furnace. It has been nearly completely overcome by improving the lines of the furnace, making the walls thin, and using water cooling. Such long-period irregularity, however, is rather different from the short-time variations which we have been considering.

Temperature of the Blast.—When blast furnaces were blown by cold air the variations in its *temperature* as it entered the furnace from, say, 0° C. to 40° C. (32° Fahr. to 100° Fahr.), made little difference to the running of the fuurnace; it certainly made much less difference than the difference in the quantity pumped by the engines, resulting from the difference in temperature of the outside air. But that is another consideration, and we can say of the temperature of the blast, considering its *temperature only*, that a difference of 40° C. would not make more than 40° C. difference in the furnace temperature, and therefore introduce a rather small irregularity. When hot blast was first used, the iron pipe stoves, operating continuously, furnished blast of fairly constant temperature, with unimportant variations. When, however, regenerative fire-brick stoves came into use, it was soon evident that the discontinuous principle of operation caused great variations in the temperature of the blast, which would be highly heated when first turned onto a hot stove, and have its temperature fall several hundred degrees by the time the stove was turned off blast. This difficulty was met by increasing the number of stoves from two to three, four, and even five, and thus correspondingly reducing the time that any one stove received blast. The uniformity of high blast temperature thus attained has been satisfactory, but not entirely so. "Equalizers" were next invented, to stand near the furnace to receive the hot blast from the stoves and to smooth down the "peaks" and raise the "valleys" in its temperature. They are gigantic thermal flywheels. Fine results have been thus achieved, and splendid uniformity of blast temperature has correspondingly increased the uniformity of the furnace working.

Moisture of the Blast.—Nature varies the atmospheric conditions so that one day the air may be very dry and the next very moist; close observation shows large variations at times even from hour to hour. But increase in moisture cools off the furnace and renders the working irregular. The manager could meet these conditions by varying the amount of fuel in the charge, if he knew 24 hours in advance how much moisture he had to prepare for, since it takes about that time for a fresh charge to descend to the foot of the furnace. That being impossible, he tries to meet the unfavorable conditions by increasing the temperature of the blast. That keeps him "on the jump"; it is meeting one irregularity by practising another, and is never more than moderately successful.

Mr. Gayley first practically solved this problem by drying the blast, reducing its moisture at all times to a small, substantially uniform amount. He achieved success, and his improvement is notable in two ways, viz., it reduces the absolute amount of moisture, thus effecting material saving in fuel, and it leaves it uniform in amount, thus abolishing one of the most annoying and serious causes of furnace irregularity. To have the moisture *uniform* is a great advantage; to have it *uniformly low* is the splendid combination which works the wonders which Mr. Gayley has achieved.

Quality of the Blast.—Everyone skilled in blast-furnace art knows that the more air is pumped into a furnace the faster it runs. Shut the air off, and it stops working altogether. Regularity of working, at uniform rate, is therefore absolutely dependent upon the furnace receiving a uniform weight of oxygen per minute.

This depends absolutely on two factors, the speed of the blowing engine and the temperature of the air entering its cylinders. Make these uniform, and uniform rate of running of the furnace results.

Mr. Gayley has attacked this problem also. In his United States patent 935,628, of September 28, 1909, he describes appliances for furnishing the blowing engine with air of substantially uniform temperature. Where the engine is distant from his drying apparatus (which also cools the air) he passes it to the engine through pipes or conduits with heat insulating sides, protecting it from outside sources of variation of temperature. so that the temperature furnished to the engine shall be constant.

Mr. Gayley avers that he is the discoverer of the great importance of the substantially uniform temperature of the air as it enters the blowing engine. It is now in order for some one to "discover" that it is of equally great importance for the *speed* of said blowing engine to be kept substantially uniform—of which equally important fact there is no hint or mention, suggestion or suspicion in the aforesaid patent, specification or claims.

It seems to the writer that variations in the temperature of the air as it comes to the blowing engine will always be present, that absolute uniformity in this respect cannot be practically attained, and that a much more practical solution would be the following: Construct a mechanical thermo regulator controlled by a thermometer, which will increase the speed of the engine 1 per cent for every 3° C. rise in temperature of the incoming air (1 per cent for every 5° Fahr. rise), and the problem of supplying the furnace with a constant weight of air is solved with exactness.

The Preparation of Silicon in the Laboratory.

BY SAMUEL A. TUCKER.

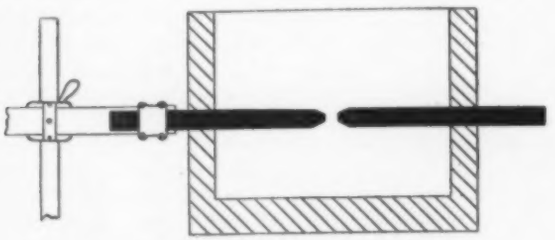
The reduction of silica by carbon in the electric furnace is now carried on quite extensively on a large scale, but its preparation in the laboratory has presented certain difficulties. The method here described can be worked very successfully and affords an interesting application of the electric furnace.

For the charge 12 kilos of a mixture of sea sand and gas coke in the proportion of 77 parts of sand to 25 of coke are used, and placed in a rectangular furnace of firebrick giving a working space of about 10 in. x 9 in., by 8 in. high, which

should be cemented as tight as possible to prevent the admission of air. At a height of 3 in. from the bottom at each end openings are provided for the two horizontal graphite electrodes, 2 in. diameter. These electrodes are adjustable so that at the start the two ends are in contact, but as the operation proceeds they can be separated about 1½ in. to 2 in.

Alternating current was used in the experiments here described and the capacity available was about 50 kw, with means for regulation of the voltage and an impedance coil to balance the arc at the start. After a minute or so the impedance coil was cut out of the circuit and the arc maintained at the terminals of the alternator.

The charge being well mixed and ground so as to pass, say, a 10-mesh screen, is placed in the furnace to a depth of 3½ in. The arc can then be started by withdrawing the electrodes and more of the charge added, carefully at first so as not to ex-



LABORATORY FURNACE FOR PRODUCTION OF SILICON.

tinguish the arc, and then the rest of it so that the furnace is completely filled. The electrodes can then be given a little separation so that a longer arc is obtained and the process carried on in this way to the end of the run.

The first run was as follows:

Time, Min.	Amperes.	Volts.
Start— 3	300	54
13	400	65
28	580	30
38	650	31
46	500	36
53	500	48
63	500	48
64	500	54
71	360	58
81	300	..

The gases find a vent and come off at considerable pressure and produce a great deal of vapor, so that good ventilation of the room in which these experiments are conducted is advisable.

After the furnace has cooled down it will be found that a crater has formed about the arc and that approximately one-quarter of the charge has been used up. The silicon is found on the bottom and in the above run the yield was 200 grams, having all the characteristics of the commercial metallic silicon. The next run was made with the same furnace and charge, but the energy was greater, as the following figures show:

Time, Min.	Amperes.	Volts.
Start—15	220	57
35	550	39
60	590	52
63	660	55
68	670	58
70	680	64
74	660	69
78	700	69
82	740	71
85	700	72

The yield in this run was 490 grams, but the heat was somewhat too great for the size of the furnace, as the brick work was melted in places and ran down with the silicon so that the specimen obtained was not quite so clear as in the first case. For the size of furnace used about 20 kw-hours would be about right.

Electrochemical Laboratory,
Columbia University.

Boiler Feed Waters with Special Reference to Treatment.

By E. G. Bashore.

Water, whether it falls to earth as rain or snow, is never pure. Rain water contains dissolved or absorbed gases—carbon dioxide (CO_2), oxygen (O), ammonia (NH_3) and during thunder showers, nitric acid. Precipitated moisture either in the form of ground water or surface water accomplishes a marked errosive action on the earth's crust, both through mechanical weathering and chemical action. The result of the former type is shown by matter in suspension, while the latter manifests itself through the dissolved impurities, and usually both processes occur simultaneously and mutually aid each other. By means of seeping water, easily soluble rocks, salts and minerals are dissolved as rock salt (NaCl), potassium chloride (KCl), magnesium chloride ($MgCl_2$), potassium carbonate (K_2CO_3), sodium sulphate (Na_2SO_4), potassium sulphate (K_2SO_4), sodium carbonate (Na_2CO_3) and gypsum ($CaSO_4$).

Many rocks and minerals which are sparingly soluble in pure water are put into solution in large quantities through the action of water containing carbonic acid (CO_2). Prominent in this class are the carbonates, calcite ($CaCO_3$), dolomite ($CaMg(CO_3)2$), magnesite ($MgCO_3$) and spathic iron ore ($FeCO_3$). Silicous rocks, such as hornblende (containing silica hydrate, with aluminium oxide, magnesium oxide, calcium oxide, soda, and usually some iron) may be decomposed by water containing carbon dioxide and certain component parts held in solution as bicarbonates of potassium, sodium, calcium, magnesium, etc. Furthermore, water may be contaminated with free acid. Such pollution may be due to organic vegetable acids from swamps or bogs, acid discharge from industrial plants or often drainage from coal or ore mines, sulphuric acid being prevalent where the ores are sulphides.

The importance then that the nature of a boiler-feed water bears to boiler operation may be made obvious by considering the quantity of water evaporated with its consequent residual products. For instance, in a 100-hp boiler using water containing only 7 grains of solids per gallon, over 300 lb. will be the quantity of solid matter for one month, while many spring waters would give over 2000 lb. of residual matter, which must either be blown out or deposited.

As a result of the impurities that a feed water may carry, the following conditions are conventionally recognized in boiler practice, namely, scale formation, corrosion, foaming and priming.

Many reagents have been suggested for the treatment of feed waters, prominent among which are lime, soda ash, caustic soda, barium hydrate, barium chloride, tri-sodium phosphate, petroleum, tannate of soda and tannin extract. Oak, hemlock, summach, logwood, etc., are used for scale on account of tannic acid being extracted; also vinegar, fruits, molasses, distillery refuse, etc., have been used for the prevention or removal of scale, their action depending upon the presence or formation of acetic acid. Organic decoctions of the type are to be avoided, for often they are injurious to the iron and cause corrosion and pitting. No absolute rule can be established for the treatment of all feed waters, nor can a "cure-all" be improvised, as each feed water from a different source will demand a treatment peculiar to itself.

The functions of the chemical treatment of a boiler feed water is to minimize the existing evil. However, it must be remembered that every chemical reaction calls for a by-product, and while possible, through chemical treatment to

correct one unfavorable condition, it is also possible to induce another. But with intelligent operation, and through cognizance of the principles involved, a great saving in coal and labor in cleaning the boiler can often be accomplished in industrial plants through the application of a properly advised treatment.

The processes for conducting the treatment can be outlined under three heads:

(1) The addition of some substance in the water in the boiler;

(2) Preliminary treatment with heat in some special apparatus;

(3) Preliminary chemical treatment.

In considering the first subhead (1), emphasis should be made of the fact that a feed water should always be purified before it enters the boiler, for the pertinent function of a boiler is to generate steam and not to serve as a still in which to conduct chemical reactions. However, cases arise where it might seem advisable to use some "solvent" in the boiler for the purpose of converting the scale into a form in which it will be easily dislodged. Such "solvents" (usually soda ash, sodium hydroxide or tri-sodium phosphate) should be used with a thorough understanding of their reactions. Too great concentration should be guarded against on account of priming; the danger of severe foaming if animal or vegetable oil is present in the water on account of saponification should be borne in mind; and great care should be exercised that the dislodged scale does not cause overhauling and burning of the steel through accumulation.

Under the subhead (1) also comes the use of metallic zinc in the boiler. With waters which have a corrosive tendency on account of the presence of magnesium chloride, it has been found that metallic zinc suspended in the boiler counteracts the corrosive action. Where corrosion of iron is suspected through the agency of electrolysis, zinc is also a specific counteractant. With the use of zinc it is to be noted that the metal should be suspended at a point away from the direct heat of the furnace, and the utmost care should be taken that the zinc has perfect contact with the shell of the boiler. The use of such substances as clay, fats, starchy matters, wood extracts, vinegar, etc., in the boiler is certainly not to be recommended, and is to be avoided.

Under the second category feed-water heaters are properly classified. Aside from the economy gained by the boiler generating more steam through the use of flue gases or exhaust steam, which otherwise would go to waste, an important point is the reduction of the temporary hardness of the water. This is brought about through the liberation of the carbon dioxide at 212° Fahr., which holds the bicarbonates of calcium and magnesium in solution.

Many types of feed-water heaters are on the market, but it is hardly within the compass of this paper to describe them. However, for their successful operation it is necessary that sufficient exhaust steam be used to keep the temperature of the water to 212° Fahr if possible, and they furthermore must be kept clean.

When there is oil present in the condensed water a separator or filter should always be used.

The third sub-division, namely, purification by preliminary chemical treatment, leads to the process which will be given most consideration, for, as previously pointed out, the boiler is not a proper apparatus in which to correct the deleterious effect of bad feed water. This method is usually termed "the water softening process" and is conducted with the use of reagents in settling tanks. The simplest form of apparatus would be two tanks, one from which the treated and settled water would be drawn off for use, while the addition of reagents, agitation and settling would be going on in the other tank.

A number of reagents can be used to accomplish the precipitation of the scale-forming constitutents, but on account of the efficiency under most conditions and low cost, the lime, soda ash treatment will be considered. This process is conducted in settling tanks and is either intermittent or continuous. In the intermittent process fixed quantities of water are treated in the tanks with weighed amounts of each reagent required for the softening, while in the continuous system the requisite quantity of chemicals are introduced in direct proportion to the flow of the water. In order to insure the highest efficiency from this process, the reagents (lime and soda ash) should be added in the proper proportion and thoroughly mixed with the water, the reaction should be accelerated if possible and should be complete. The apparatus should be designed so as to cause rapid sedimentation and perfect clarification. With proper care, particularly incrustaceous waters can be softened to the extent of containing from three to five grains of incrustaceous matter per gallon.

Scale formation is due to the presence of carbonates, sulphates, sediment and mud, and soluble salts in the feed water and its deposition on the inside of boilers not only lessens the available water space but acts as a non-conductor of heat. (It is estimated that 1/16 in. of scale causes a loss of 13 per cent of fuel; ¼ in., 38 per cent, and ½ in., 60 per cent).

To calculate the amount of material that will be deposited in 1000 gal. of water per day, take the number of grains per gallon of each as shown by the analysis and divide by 7; the quotient will be pounds deposited in 1000 gal. of water per day. The scale-forming carbonates in water are those of calcium and magnesium. Both are very sparingly soluble in pure water, but the presence of carbonic acid (CO_2) in the water renders them soluble as bicarbonates; however, carbonic acid is insoluble in boiling water, and is liberated with a consequent precipitation of calcium and magnesium carbonates which tend to deposit as a soft scale. Each grain of calcium or magnesium bicarbonate reported in the analysis as carbonates, requires the number of grains divided by 5.5 expressed in pounds of lime for 1000 gal. of water per day. For example, 5.5 grains of total carbonates requires 1 lb. of lime for 1000 gal. of water per day. The precipitation of carbonates is best conducted in external settling tanks, and particularly so if the carbonates exceed 10 grains per gallon.

The sulphates which are scale forming are those of calcium and magnesium. Magnesium sulphate alone does not form a hard scale, but in the presence of calcium sulphate it forms a very stony scale hard to remove. Calcium sulphate alone forms a hard, tenacious scale. The presence of these sulphates having been determined, soda ash should be used to prevent the formation of the resultant hard scale. However, it is to be noted that the reaction forms sodium sulphate, which, if too concentrated, will produce priming. This condition will be dealt with under the proper head. The quantity of soda ash required may be calculated by dividing the number of grains per gallon of magnesium sulphate by 8 and the number of grains of calcium sulphate by 8.75, and the quotient represents pounds of soda ash per 1000 gal. of water per day. If carbonates and sulphates exceed 15 grains per gallon, external treatment is advisable.

Sediment and mud also cause scale by depositing and baking on the fire sheet of a tubular boiler, the lower tubes in a water tube boiler and the crown sheet in a marine and locomotive boiler.

Soluble salts can also be incorporated in the scale if the concentration is too great. Excess beyond 300 grains per gallon should be avoided.

Corrosion or pitting, which is by far the most dangerous condition met with in boiler practice, may be caused by free acids or substances which develop acidity.

Free sulphuric acid often exists in water and should be neutralized or precipitated. Divide the number of grains of free sulphuric acid by 6.5 and the quotient equals the number of pounds of soda ash required to neutralize 1000 gal. of

water per day. This same rule applies to hydrogen sulphide when present. Under certain conditions it is advisable to use barium chloride or hydroxide and precipitate the barium sulphate externally.

If organic acids exist in the water they have a very corrosive action and can best be corrected by external treatment with alum and soda ash. Divide the organic acidity as reported in grains per gallon in the analysis by 6 to get pounds of soda ash and by 8 to obtain pounds of alum required per 1000 gal. of water per day.

Hydrochloric acid is sometimes found in feed waters and can be corrected by the use of soda ash. The number of grains of free acid divided by 9 equals the pounds of soda ash required for 1000 gal. of water per day.

Magnesium chloride is said to be unstable at high temperatures and in the presence of heated iron and water will split up to form magnesium oxide and hydrochloric acid. The number of grains of magnesium chloride as shown by the analysis, divided by 4.8, equals the number of pounds of soda ash required for 1000 gal. of water per day.

Sodium chloride and silica are said to develop free hydrochloric acid at high temperatures through the formation of sodium silicate. To counteract this effect an alkali such as soda ash must be used. Take the number of grains of available silica present that will act on the sodium chloride and divide by 9.6 and the quotient equals the pounds of soda ash required.

Foaming and priming may be considered under the same head, for both tend to cause wet steam. Priming is due to high surface tension, while foaming is due to matter in suspension.

Sodium sulphate is found whenever soda ash is used to precipitate calcium or magnesium sulphate and is also found in many southern waters. Priming is the result of too great concentration. While the limit of concentration is determined by the type of boiler, the following figures are a close approximation of the truth. Old type two-flue boilers, 1000 grains per gallon, steam at 50 lb. pressure. Horizontal return tubular boiler, 500 to 600 grains per gallon, 100 lb. pressure. Modern water tube boilers, 250 to 400 grains per gallon, steam at 125 lb. pressure. Locomotive boilers, 100 to 200 grains per gallon, steam at 200 lb. pressure.

Sodium chloride also causes priming, but not to such a pronounced degree as sodium sulphate. Usually twice as much sodium chloride can be allowed to concentrate as sodium sulphate without producing priming.

Foaming can be caused through mud and suspended matter and the remedy is either blowing down or filtration.

Sewage and algae also cause foaming and create a scum on top of the water. This condition may be taken care of either by the use of a surface blow-off or outside filtration.

The subject as here outlined is at best only a brief resumé of conditions as met with in boiler practice with reference to the treatment of feed waters, and many cases arise where it is advisable to seek the advice of an experienced chemist or consult an engineer who makes a specialty of water softening.

Bayonne, N. J.

Decomposition Points of Some Sulphates in an Air Current.

By Walter S. Landis, M. S.

It has been rather disappointing to look in the many new treatises on chemistry and metallurgy and find such statements as "ferrous sulphate decomposes at a low red heat" or other equally vague statement. This decomposition point of the sulphates is of very great importance in all roasting operations, and now that fuel economy is being forced down our throats (because we would not take it in any other way), and the pyrometer is now as integral a part of our plant as the furnaces themselves, more exact data is wanted on such decomposition points.

The purpose of this work was to determine a few of these decomposition points of the sulphates. The work has already been done without doubt, but the results have never appeared where they are readily accessible, and have certainly not been copied in our more recent books. This determination was not made with the great accuracy that the physical chemist might desire, but rather more to simulate the actual conditions existing in a roasting furnace, that is, in a current of air such as would pass over the materials in a roasting furnace.

The method employed was to insert a porcelain boat, carrying the sulphate in a porcelain combustion tube, into a combustion furnace. Lying on the surface of the sulphate in the boat was the hot junction of a Le Chatelier thermo-couple, previously carefully standardized, the cold junction being in melting ice. Connected to the one end of the combustion tube was a three-mouthed bottle containing a solution of caustic soda carefully neutralized with sulphuric acid, using methyl orange as an indicator. The connecting tube ended in a glass tube dipping well below the surface of the liquid in the three-mouthed bottle. Through the central mouth passed the stem of a separatory funnel, in the bulb of which was placed a dilute solution of caustic soda. Finally the third mouth of the bottle was connected with an aspirator for producing the necessary air current through the apparatus. As decomposition took place the SO_3 and SO_2 passed along with the air current into the liquid in the bottle and affected the indicator. The introduction of caustic soda from the separatory funnel allowed of this acid being neutralized and served to keep the apparatus in continuous working order. Some idea, only qualitative, could be gained as to the rate of decomposition by the amount of caustic soda added to keep the indicating solution neutral.

The results obtained on the sulphates studied were as follows:

$FeSO_4$: The first traces of decomposition showed themselves at 550° C. The increase was but slight up to 580° C. and then became more rapid until 600° C. was reached when there was a sudden increase. On heating for two hours at this temperature all decomposition seemed to have stopped as only very slight traces appeared on further heating to 650° C. No further traces were noted between this last temperature and 960° C., the highest temperature attainable in the test.

$CuSO_4$: The first traces of acid appeared at 400° C. Only slight amounts were noted as the temperature was slowly increased to 690° C., when a very rapid evolution took place. A further trace was obtained at 700° C., and then no more up to 900° C., the highest temperature attained in this investigation.

$ZnSO_4$: The first traces of decomposition appeared at 730° C., more appearing at each increase of temperature up to 760° C., when the decomposition was very rapid. All the acid seemed to have been removed at this temperature for no more was found on raising the temperature to 980° C.

In all cases the experiments were repeated several times and gave almost identical temperatures for each stage at each repetition, the points seemingly very markedly characteristic of each salt.

Data of this nature needs no further comment as to its value to the roaster, showing the minimum temperature at which the furnace may be run for complete decomposition and the limits which need not be exceeded for producing the same result.

Metallurgical Laboratory,
Lehigh University.

A Primer on Explosives is to be issued by the United States Geological Survey. According to the scope, indicated in advance sheets, this primer will fill a need in aiding coal miners and all who handle explosives.

The Magnetic and Electrical Properties of the Iron-Nickel Alloys.

BY CHARLES F. BURGESS AND JAMES ASTON.

In iron and nickel we have the two most magnetic elements. The fact that a mixture of certain percentages of these two metals results in a non-magnetic alloy makes an investigation of the magnetic qualities of an entire series of these alloys of interest.

There is much literature embodying the results of the research on the irreversibility of certain properties of the nickel-iron alloys and most of the published data of magnetic investigations is concerned with the temperatures of this appearance and disappearance of the magnetic susceptibility for different percentages of nickel.

In the investigation of Barrett, Brown and Hadfield (Jour. Inst. Elec. Engrs., Vol. XXXI, page 674) there is included complete hysteresis loops for a number of nickel-iron alloys in which the carbon runs from 0.13 to 0.23 per cent and the manganese from 0.65 to 1.08 per cent. Also, the magnetization curves of a series was determined by Carpenter, Hadfield and Longmuir (7th Report of the Alloys Research Committee, Inst. Mech. Engrs., Vol. 2, 1905, page 857). In this series the carbon was that of a medium steel (0.41 to 0.52 per cent) and the manganese content varied from 0.79 to 1.03 per cent. These steels, except for the increase of carbon, were almost identical with those of Hadfield, mentioned above; in fact, the work was intended as supplementary to this previous research.

This present paper deals with an investigation of a complete series of iron-nickel alloys in which the endeavor has been to keep down the impurities to the lowest possible point in order

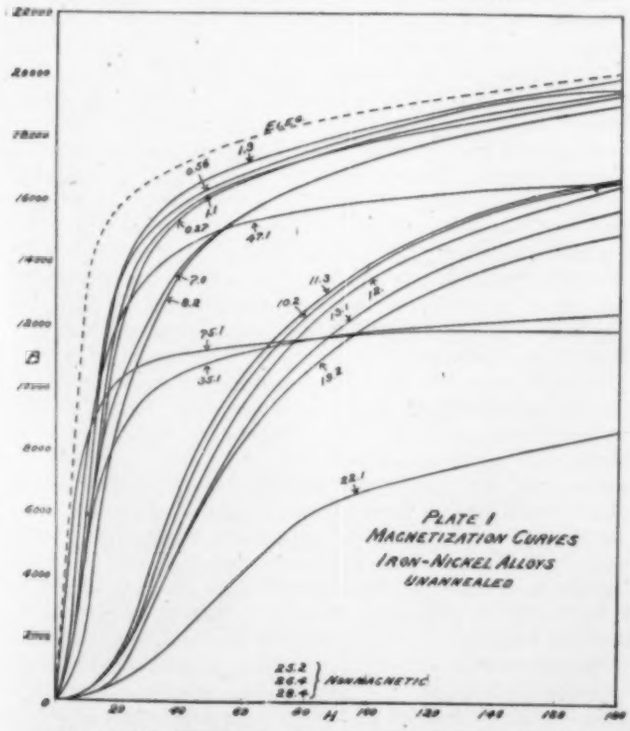


FIG. 1.—MAGNETIZATION CURVES. IRON-NICKEL ALLOYS UNANNEALED.

that the effects observed may be ascribed entirely to the alloying element. The work is part of an extended investigation of electrolytic iron and its alloys carried out during the past several years at the chemical engineering laboratories of the University of Wisconsin, under a grant of the Carnegie Institution of Washington.

The iron used is of a high degree of purity obtained by a process of double electrodeposition. The electrolytic material has a purity represented by an iron content of 99.97 per cent, the total content of sulphur, phosphorus, manganese and silicon being thus 0.03 per cent. The alloys were prepared by melting this iron with the desired quantity of nickel, also electrolytic material of a high purity, in magnesia crucibles supported by a graphite jacket. The crucibles were covered with a magnesia and a graphite lid and buried in an electric resistor furnace in which the melting was done. The resulting ingots

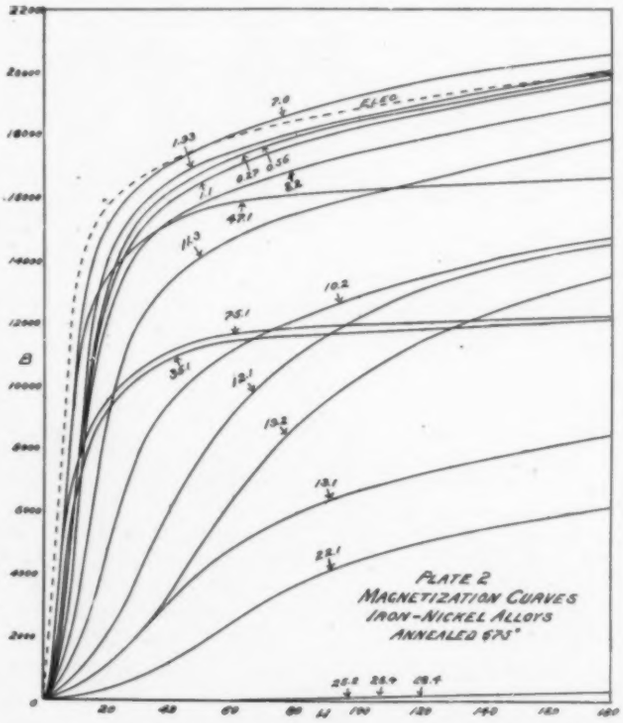


FIG. 2.—MAGNETIZATION CURVES. IRON-NICKEL ALLOYS ANNEALED 675°.

of about 500 grams (1 lb.) in weight, were heated in a forge fire and drawn down under a steam hammer into rods about ⅝ in. in diameter and 18 in. long.

Experience has shown that, in spite of every precaution, there is a slight carbon absorption, unquestionably due to the reaction of the iron upon the carbon monoxide atmosphere of the furnace. The amount, however, is small, and may be assumed in most cases to be well under 0.10 per cent. The other impurities, such as silicon, sulphur, phosphorus and manganese, are entirely negligible. In the present discussion the materials may be considered as binary alloys of iron and nickel and the effects to be due to these two elements alone.

Analysis.

The following series was subjected to magnetic tests.

Bar.	% Nickel— Added.	Analysis.	% Carbon.
144C	0.25	0.27	
144D	0.50	0.56	
144E	1.0	1.07	
144F	2.0	1.93	
144J	7.0	7.05	
157A	8.0	8.17	
157D	10.0	10.20	
157E	11.0	11.29	
157H	12.0	12.07	
144M	13.0	13.11	0.088
144P	19.0	19.21	
166G	22.0	22.11	
154S	25.0	25.20	
166I	26.0	26.40	
166C	28.0	28.42	
166L	35.0	35.09	
166O	50.0	47.08	
166Q	75.0	75.06	

As will be seen, iron and nickel alloy readily and without segregation, since the agreement is strikingly close between the added percentage and that found upon analysis. This evidence is more conclusive when it is remembered that the analyses

were made from turnings during the machining of test bars and after the forging of the ingot.

From the rods forged from the ingots, bars were reserved for tensile strength and one was taken for the magnetic tests. The latter was finished to a diameter of 1 cm with a length of about 20 cm. No sample of pure nickel was tested, since it was impossible to forge the ingot into a bar. It was possible to turn all bars in the lathe, although there was considerable difficulty in the range from 10 per cent to 20 per cent of nickel because of the hardness.

Magnetic Tests.

The magnetic tests were made upon an Esterline permeameter. This instrument is essentially a small generator, the armature of which is driven at a predetermined constant speed. The voltage generated is proportional to the magnetic flux through the armature, since all factors are constant except this flux. The total number of lines of force is equal to that in the bar under test, since this bar forms part of the magnetic circuit. The necessary rheostats and switches are provided for regulation and reversal of the magnetizing current, for compensation, etc. But one recording instrument is used—a Weston millivoltmeter, calibrated to read directly, without calculation, the flux density B and the magnetizing force H in C.G.S. units.

This instrument was selected because of its adaptability to the work on the several hundred bars on hand and in prospect. It has given good satisfaction in its present form, both as regards rapidity and consistency. The results are not based upon an absolute measurement and the instrument must be calibrated against a standard determined by ballistic test. In our tests, magnetization curves were taken by the method of reversals and hysteresis loops by the step-by-step process. Complete magnetization curves and readings of the retentivity and the coercive force were taken under four different heat treatments, first, as forged; second, after heating to 675 deg. C and slowly cooling; third, after heating to 1000 deg. and slowly cooling, and fourth, after heating to 900 deg. and quenching in water.

Table I. Data of Magnetic Tests.

Bar	% Nickel	H = 50 Unann	H = 50 Ann. 675°	H = 50 Ann. 1,000°	H = 50 Quench 900°	H = 20 Unann	H = 20 Ann. 675°	H = 20 Ann. 1,000°	H = 20 Quench 900°	H = 10 Unann	H = 10 Ann. 675°	H = 10 Ann. 1,000°	H = 10 Quench 900°	H = 100 Unann	H = 100 Ann. 675°	H = 100 Ann. 1,000°	H = 100 Quench 900°
144C	0.27	2,600	6,000	9,100	10,450	11,400	13,200	13,650	13,900	16,150	16,850	16,800	16,750	17,900	18,400	18,300	18,100
144D	0.56	5,900	5,100	11,950	12,000	13,250	13,250	14,800	15,000	16,400	16,750	17,000	18,050	18,200	18,500	18,400	18,400
144E	1.07	3,500	5,100	11,500	7,600	12,050	12,800	14,650	11,200	16,250	16,500	17,000	15,600	17,800	18,250	18,400	18,400
144F	1.93	4,850	6,600	10,600	9,700	13,250	13,950	14,800	14,200	16,750	17,050	17,000	16,950	18,300	18,550	18,400	18,400
144J	7.05	4,850	10,000	9,500	9,300	9,950	15,050	13,800	13,250	14,750	17,550	17,000	16,750	17,250	19,250	18,750	18,500
157A	8.17	2,600	5,100	8,000	8,350	8,750	12,050	12,150	13,000	14,800	15,800	16,250	16,750	17,250	17,350	18,300	18,500
157D	10.20	50	1,450	5,300	7,950	900	4,650	10,150	12,350	8,550	10,500	15,650	16,450	13,800	12,850	18,350	18,600
157E	11.29	150	2,800	4,100	5,600	1,300	8,800	7,650	10,100	8,900	14,150	12,400	15,450	13,950	16,000	15,400	18,050
157H	12.07	350	750	2,700	4,600	1,300	2,000	5,900	10,100	7,700	7,750	10,700	15,250	13,450	12,200	13,850	17,700
144M	13.11	100	450	700	2,450	800	1,150	1,850	8,150	6,850	4,100	6,800	14,600	12,600	6,650	11,550	16,650
144P	19.21	400	400	650	2,200	1,150	1,050	1,700	7,000	6,750	5,000	7,050	12,950	12,050	10,350	12,050	17,200
166G	22.11	100	100	400	300	500	350	1,100	800	3,100	1,800	4,800	4,150	6,800	4,500	8,500	8,050
154S	25.20	0	0	50	50	0	0	150	100	0	0	500	350	0	100	1,050	850
166I	26.40	0	0	100	0	0	0	200	50	0	0	650	350	0	0	1,200	850
166C	28.42	0	0	0	0	0	0	0	0	0	0	0	0	0	100	100	100
166L	35.09	4,850	6,400	8,800	6,500	8,350	9,250	10,750	8,400	10,850	11,300	12,000	10,650	11,850	11,700	12,300	11,800
166O	47.08	7,850	10,600	13,650	10,300	12,100	13,450	15,150	12,400	14,950	15,550	16,250	14,800	16,000	16,200	16,500	16,000
166Q	75.06	8,300	7,200	8,450	8,700	10,200	9,550	10,650	11,200	11,250	11,450	11,850	11,800	11,800	11,950	12,300	12,100

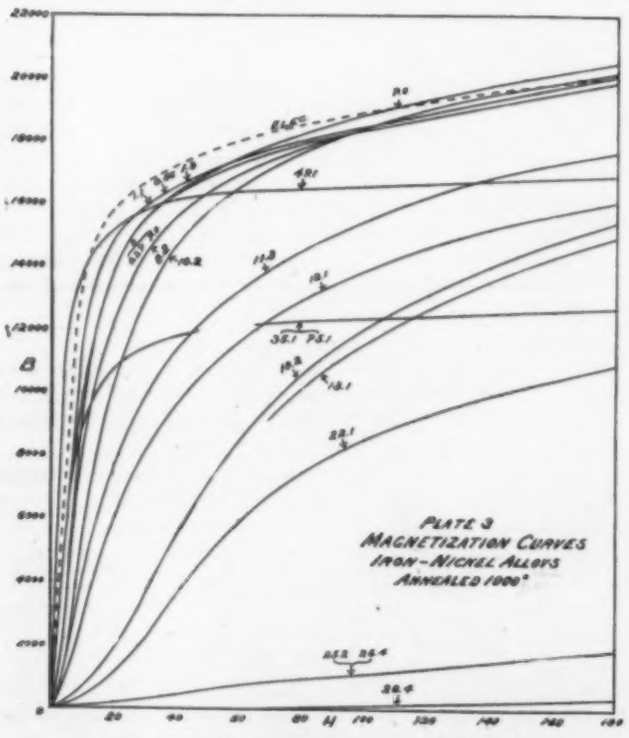


FIG. 3.—MAGNETIZATION CURVES. IRON-NICKEL ALLOYS ANNEALED 1000°.

The results of the magnetic tests are given in Tables I and II and in the curves of Plates 1, 2, 3 and 4. For purposes of comparison, there is given in each plate a dotted curve, plotted from the data of a test on a bar of electrolytic iron in the unannealed state, and which is of very high magnetic quality. The effect of the increase in the percentage of nickel is very apparent. In the unannealed state the alloys fall into several groups. Bars 144C, 144D, 144E and 144F, with 0.27, 0.56, 1.07 and 1.93 per cent of nickel, are very close together, and but little inferior to the standard. Increase of the nickel to 7.05 and 8.17 in bars 144J and 157A, has resulted in a falling off in quality. This is still greater for 157D (10.20 per cent), 157E (11.29 per cent), 157A (12.07 per cent), 144M (13.11 per cent) and 144P (19.21 per cent), which have a low permeability, and the curves of which take positions in the order of the nickel content. The quality of bar 166G, with 22.11 per cent, has become very poor, and finally bars 154S (25.20 per cent), 166I (26.40 per cent) and 166C (28.42 per cent), have become non-magnetic. Further increase of nickel content in 166L, 166O and 166Q, with 35.09, 49.08 and 75.06, has restored the magnetic quality very markedly, the maximum values being reached with 166O (47.08 Ni). Below the saturation bend of the curves (at about H = 40) these three bars have a very high permeability, but beyond this point the permeability becomes practically unity, with the flux density increasing but little more than the increase in the value of the magnetizing force. This latter feature is even more marked after the subsequent annealings at 675 deg. and 1000 deg.

Annealing at 675 deg. has resulted in general in an improvement of magnetic quality throughout the entire range. This is most marked for 144J, with 7.05 per cent Ni, and 157E, with 11.29 per cent Ni. On the contrary, there has been a deterioration in bars 144M (13.11 per cent), 144P (19.21 per cent) and 166G (22.11 per cent); this is very pronounced in the case of the first of these. A very slight amount of magnetism has been induced in the three non-magnetic bars 154S, 166I and 166C.

The second annealing at 1000 deg. has for those bars with smaller nickel content resulted in an increase of permeability for the lower magnetizing forces, with but little change in the upper ranges. Very pronounced changes have taken place in

bar 157D, with 10.20 per cent Ni, which is better throughout, as are bars 157H (12.0 per cent), 144M (13.11 per cent) 144P (19.21 per cent) and 166G (22.11 per cent). The permeability of bar 166O, with 47.08 Ni, has become very high at the lower values of H; in fact, it is about the highest of any of the materials tested in this work. Bars 154S (25.20 per cent) and 166O (26.40 per cent) have become appreciably magnetic.

Quenching at 900 deg. has not resulted in much change, except for the alloys with 11.29 per cent, 12.07 per cent, 13.11 per cent and 19.21 per cent of nickel (157E, 157H, 144M and 144P) where the permeability has greatly increased.

Referring to Table II, the retentivities tend to decrease as we increase the nickel to the compositions of the non-magnetic alloys; and the values of coercive force increase. Beyond this composition the retentivities are again evident, and the coercive forces are small. This is in line with the deductions drawn from the curves and data. In general the coercive forces become smaller with each succeeding heat treatment. For the smaller nickel percentages of 0.27 and 0.56, in bars 144C and 144D, the values 4.3 and 4.2 are about those of electrolytic iron;

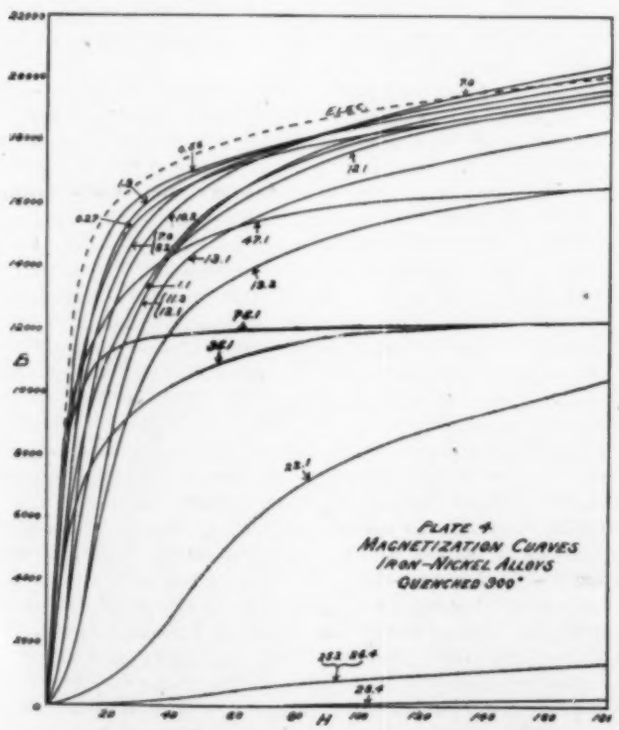


FIG. 4.—MAGNETIZATION CURVES. IRON-NICKEL ALLOYS QUENCHED 900°.

and the values of coercive force for the three bars with the maximum content of 35.08, 47.08 and 75.06 per cent are markedly low.

The alloys with medium percentages of nickel (10 per cent to 20 per cent) seem to be very susceptible to heat treatment, and are at their best after quenching. This might be explained from the fact that nickel has a tendency to lower the temperature of transition in the iron from the hard γ to the soft α form. Consequently, while slow cooling would result in γ state, with its hardness and low permeability, the quenching, with cooling at lower temperatures, might have a tendency toward transition to the α form, with a resultant softness and better magnetic quality.

The strange feature of this nickel-iron series is that with two highly magnetic materials we can obtain all degrees of magnetization, even to a non-magnetic range. Equally striking is the fact (not here investigated) that these alloys in the non-magnetic range at normal temperatures become strongly magnetic at lower temperatures which are dependent upon the composition. They remain magnetic upon a rise of temperature of several hundred degrees, when they again become non-magnetic. Many attempts have been made to explain this anomaly, the reasons being based largely upon allotropy in the two elements.

The behavior is such that one might almost suppose that in the molecular arrangement of the alloys the iron and nickel

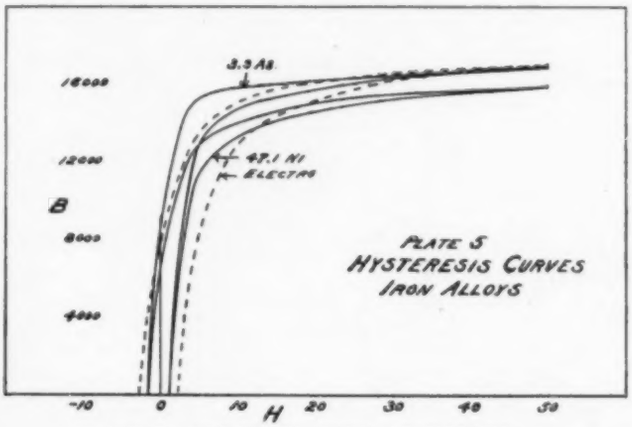


FIG. 5.—HYSTERESIS CURVES. IRON-ALLOYS.

become so oriented as to be of opposing polarity; this opposition becoming more pronounced with increase of the nickel content until finally a balance is reached with the resulting non-magnetic alloy. A further increase of nickel upsets this balance in favor of the nickel with its polarity preponderating.

In the range of commercial usage with 2 to 4 per cent of nickel the magnetic properties of the alloys have not fallen off to such an extent as to obviate against the use of the nickel in case the resulting mechanical qualities required are of sufficient importance to warrant its selection.

Hysteresis.

In connection with the magnetic tests, hysteresis curves were taken on such alloys as showed high permeability, especially such as had low magnetizing forces, and in consequence seemed particularly adapted to transformer working.

In the tests the cycle was carried to a maximum magnetizing force of H = 50. Typical bars with low hysteresis losses are noted in the following table; all were annealed at 1000 deg C.

Bar.	Per Cent Composition.	Relative Area of Loops.
117A	Electroforged	1.00
NSW	Swedish anode	1.08
55D	0.067 Al.	1.10
113D	3.862 As.	0.58
123C	1.035 Co.	1.20
166O	47.080 Ni.	0.53
96H	4.655 Si.	0.76
121J	2.059 Sn.	1.01

The results of the measurements upon bar 166O (47.08 per cent Ni) are plotted in Plate 5, together with those on electrolytic iron bar 117A, which was used as the standard of comparison, and 113D with 3.86 arsenic, which latter material showed the highest magnetic quality of any sample tested. The half-loops only are given.

In the above table is noted the relative area of the loops compared to the standard 117A as unity. These figures are indicative of the relative hysteresis losses per cycle of the materials in question. In our tests, therefore, the 50 per cent nickel alloy had the lowest loss, but 53 per cent of the standard and exceeded even the arsenic bar where the loss was 58 per cent of the standard. However, the great cost of this large quantity of nickel would eliminate its extensive use in general practice; also it is in reality not so good a material as some of the other alloys which have but sightly higher loss per cycle and which reach values of flux density considerably higher than the nickel bar for the same magnetizing forces.

Electrical Resistance.

As a study concomitant with that of the magnetic properties

the electrical resistance of the bars of the nickel-iron series was determined. The measurements were taken by the fall of potential method by recording the drop across a measured

Table II. Coercive Force and Retentivity.

		H (Max.) = 200				H (Max.) = 200			
		Unann	Ann. 675°	Ann. 1,000°	Quench 900°	Unann	Ann. 675°	Ann. 1,000°	Quench 900°
		Coercive force				Retentivity			
Bar	% Nickel								
144C	0.27	13.4	9.0	6.0	4.3	12,400	12,500	10,600	10,900
144D	0.56	11.0	9.1	4.0	4.2	11,400	12,300	10,100	10,700
144E	1.07	11.4	9.3	4.9	6.0	12,000	12,800	11,300	8,900
144F	1.93	10.2	9.0	5.0	6.0	12,800	14,400	11,300	11,700
144J	7.05	11.0	7.5	5.8	5.5	9,100	14,200	10,100	10,700
157A	8.17	12.5	10.0	6.3	6.0	8,000	12,500	9,600	9,500
157D	10.20	44.0	18.0	8.7	7.2	8,700	9,500	8,600	10,300
157E	11.29	32.5	13.5	9.2	8.8	9,200	11,800	7,000	9,500
157H	12.07	35.0	35.5	10.7	9.5	8,800	9,800	5,900	9,500
144M	13.11	34.8	32.5	31.5	14.9	8,500	4,500	7,700	11,900
144P	19.21	38.5	40.5	34.2	15.5	8,400	8,300	8,500	9,300
166G	22.11	44.5	51.8	37.0	42.7	5,000	3,900	6,300	6,100
154S	25.20	Non.	200+	50.3	64.5	Non.	200	1,000	1,000
161I	26.40	Non.	200+	45.5	66.3	Non.	200	1,100	900
166C	28.42	Non.	200+	200+	200+	Non.	2,000	400	400
166L	35.09	7.9	6.5	1.9	2.7	4,800	6,200	4,000	4,500
166O	47.08	6.7	5.5	1.6	1.9	7,700	9,000	8,700	5,200
166Q	75.06	3.1	3.5	1.9	1.5	6,000	4,600	3,600	4,200

length of the bar when a large current of about 50 amp was flowing. In the following table the results are given in

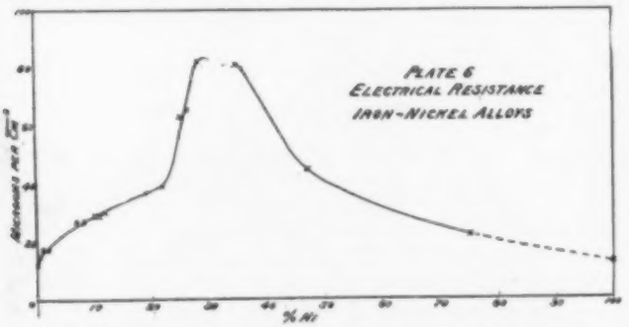


FIG. 6.—ELECTRICAL RESISTANCE. IRON-NICKEL ALLOYS.

microhms per cubic centimeter. There is also noted the relative resistance in terms of the same standard bar of the electrolytic iron which was used in the magnetic tests. (See also Fig. 6.)

Electrical Resistance.

Bar.	% Ni.	Microhms per cu. cm.	Relative resistance.
117A	0	12.1	1.00
144C	0.27	13.1	1.08
144D	0.56	15.4	1.27
144E	1.07	16.9	1.40
144F	1.93	16.4	1.36
144J	7.05	26.9	2.22
157A	8.17	26.7	2.20
157D	10.20	28.6	2.36
157E	11.29	29.4	2.43
157H	12.07	30.3	2.50
144M	13.11	34.8	2.62
144P	19.21	36.2	2.99
166G	22.11	38.7	3.20
154S	25.20	63.2	5.22
166I	26.40	65.5	5.41
166C	28.42	82.0	6.77
166L	35.09	81.1	6.70
166O	47.08	44.7	3.69
166Q	75.06	22.1	1.83
	100.00	12.4	1.02

There is a gradual increase of resistance with the increase of nickel up to a percentage of 22.11, where there is a sudden rise in the curve at compositions corresponding to those of the non-magnetic alloys. However, the curve continues to rise, reaching a maximum in the range between 28.42 per cent and 35.09 per cent. At these two points the resistances are respectively 82.0 and 81.1 microhms per c.cm, or 6.77 and 6.70 times the resistance of the standard. Beyond this range the two determinations made indicate a fall of resistance with increase of nickel in a curve very similar to that on the other side of the maximum range. The final point of the curve at a nickel content of 100 per cent is taken from a standard handbook. Might not the maximum point in this curve of resistance be at 34 per cent of nickel, corresponding to the compound Fe_2Ni? Measurements of other kinds, such as dilatation, indicate a change of properties at this composition and tend to support the above view.

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Current Densities and Energy Losses in Electrodes.

By Jakob Forssell, Ph.D.

At the meeting of the American Electrochemical Society in Philadelphia, May, 1907, a paper on "carbons for electrometallurgy"[1] was read by F. A. J. FitzGerald and the writer. It was our intention at that time to continue our investigation of this subject in order to obtain a full knowledge of the influence and the relations of the different properties of carbon electrodes with regard to energy losses and current densities in electrodes. The paper by Mr. FitzGerald on "the heat conductivity of carbon"[2] read at the meeting of the society in New York, October, 1907, was another contribution to the series of researches contemplated.

In order to obtain a basis for further work along these lines the writer of this article attempted at that time to carry out a theoretical analysis of the different factors influencing the energy losses in electric furnace electrodes. The results obtained from this analysis seemed to offer a very promising guide for the experimental work. However, for various reasons this work was interrupted. The general conclusions derived from the theoretical treatment of the subject as given below were demonstrated and discussed in a paper on "some principles regarding the construction of electric furnaces" read by the writer on Oct. 8, 1908, before the Chemical and Metallurgical Section of the Swedish Association of Technical Societies in Stockholm.

Now that the work on this subject has been taken up again by Mr. C. A. Hansen[3], it seems to the writer that a publication of his calculations may be of interest, since the results obtained by Mr. Hansen appear as logical consequences of these theoretical calculations, thus tending to confirm their correctness. The value of this analysis of the problem regarding energy losses and current densities in electrodes, however, extends far beyond an interpretation of this special case. It gives a general understanding of the relations of the different factors influencing the working of furnace electrodes under different conditions and therefore may serve as a useful aid in the construction of electric furnaces and in suggesting further valuable and necessary experiments regarding electrodes.

In dealing with the problems of energy losses in furnace electrodes it will be necessary to consider the conditions in furnaces of different types. We will, therefore, first take up the question with regard to such furnaces where the electrode enters the furnace through an insulating medium or wall, as shown in Fig. 1. This is the case, for instance, in a number of resistance furnaces.

The general arrangement assumed is evident from the cut. The energy waste consists of two main parts, i. e., heat generated in the electrode by the electric current and heat conducted away from the charge into the electrode. It is disposed of by conduction to the cable connection and by the cooling influence of the atmosphere surrounding the protruding part of the electrode. Further, some heat will also be conducted away by the wall surrounding the electrode.

In order to carry out an exact analysis of the energy losses in the terminal, we would have to know, not only all its dimensions and working conditions, but also the mathematical expressions for the electric resistivities of the materials of the electrode and of the surrounding wall and for the thermal conductivities of said materials as functions of the temperature. Even if we possessed such a knowledge the deduction in this way of the energy losses incurred would lead to a very complicated expression, unhandy for practical use and undoubtedly requiring empirical modifications in its practical applications. It, therefore, seems more rational to carry out our first analysis while assuming certain simplified conditions and to introduce afterward such corrections as are required.

[1] Trans. of Am. Electrochem. Soc., Vol 11, p. 317.
[2] Trans. of Am. Electrochem. Soc., Vol. 12, p. 165.
[3] Trans. of Am Electrochem. Soc., Vol. 15, p. 279.

The assumptions made are:

1°. No exchange of heat takes place between the electrode and the surrounding wall.

2°. The electric current is confined to the electrode, no leakage into the surrounding wall taking place.

3°. The electric resistivity and the heat conductivity of the electrode are constant and independent of the variations of temperature.

4°. A uniform temperature prevails in each cross-section of the electrode.

We will use the following notation:

A = area of cross-section of terminal in sq. cm.

I = current in amperes.

a = length of terminal inside of the furnace in cm. (from E to B in Fig. 1).

l = length of terminal outside the furnace in cm. (from B to C).

d = current density in amperes per sq. cm..

t_1 = temperature of terminal in a section through E.

t_2 = temperature of terminal in a section through B.

T = temperature drop between E and B $(= t_1 - t_2)$.

Q = heat conducted from charge through the section at E into the electrode (in watts).

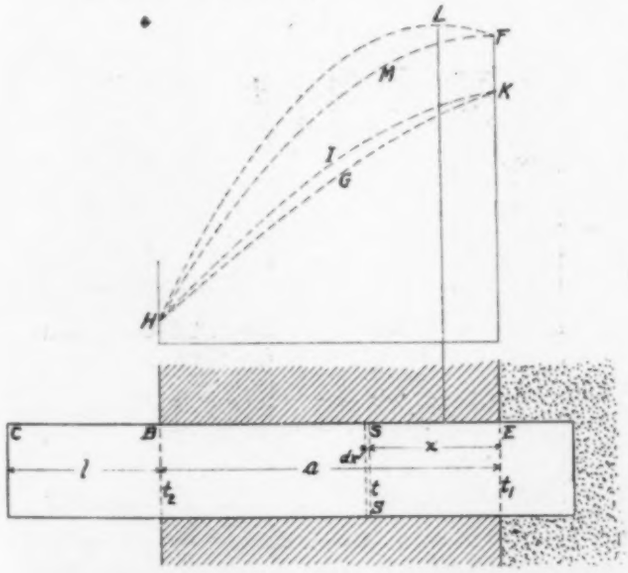


FIG. 1.—HEAT FLOW AND TEMPERATURE DROP IN FURNACE ELECTRODES.

W_1 = power in watts lost by heat conduction from the charge and on account of the electrical resistance of the terminal between E and B.

W_2 = power in watts lost on account of the electrical resistance of the outside part (from B to C) of the terminal.

h = heat conductivity of terminal.

r = electrical resistivity of terminal (per cm. cube).

k = heat equivalent of one watt (= 0.24 cal.).

The variables t and x signify the temperature t in a certain cross-section of the electrode at a distance x from the inside of the wall (E).

We will regard the quantity of heat passing through an imaginary, very thin slice of the terminal at the cross-section $S-S$. The thickness of this slice is dx and the temperature difference between its sides dt. The heat passing through it consists of the heat lost from the charge ($= kQ$) and the heat produced by the electric current from S to E ($= krxI^2/A$). Since the temperature difference between the sides of the slice, its thickness and area are known, the heat passing must also be equal $-hAdt/dx$. We obtain consequently the following differential equation:

$$-hAdt/dx = kQ + krxI^2/A \qquad (1)$$

By integration and from the facts that

$$t = t_2 \text{ for } x = a$$

and

$$t = t_1 \text{ for } x = o$$

and further

$$t_1 - t_2 = T$$

we obtain:

$$hA\,(t_1 - t) = (hTA/a - \tfrac{1}{2}\,kraI^2/A)\,x + \tfrac{1}{2}\,krI^2x^2/A \qquad (2)$$

$$Q = hTA/(ka) - \tfrac{1}{2}\,raI^2/A \qquad (3)$$

Equation (2) gives the temperature t in any section at a distance x from E. It represents a parabola with vertical axis.

The temperature drop per cm. (dt/dx) in any such section we obtain by derivation of equation (2):

$$-hAdt/dx = hTA/a - krI^2(a/2 - x)/A \qquad (4)$$

The left-hand term expresses the amount of heat passing through a given section at the distance x from E. The total loss of energy in watts in the part of the terminal inside the wall (E to B) by conduction from the charge and from the heat generated by the current we find by inserting a for x. We thus obtain:

$$W_1 = hTA/(ka) + \tfrac{1}{2}\,raI^2/A \qquad (5)$$

If no electric current was flowing through the electrode, the energy lost by conduction of heat from the charge into the electrode would amount to $hTA/(ka)$ watts, but on account of the current heating the terminal, the energy lost by conduction from the charge into the electrode is decreased by an amount equal to one-half of the heat generated by the current in the terminal. Thus we find that the heat generated by the current serves a useful purpose in keeping down the loss from the charge.

We will now consider the influence of the assumptions made in these deductions. According to our first assumption no exchange of heat should take place between the electrode and the surrounding wall. This condition will usually not be fully satisfied in practical working. However, as this subject will later be discussed at some length, it may at present be sufficient to say that the corrections to be made on this account will, as a rule, not affect our results very materially.

Regarding our second assumption that the current should be confined to the electrode, no leakage through the wall taking place, it probably in most cases answers the conditions in practical working so nearly that no corrections on this account are needed. One point should be raised, however, in this connection concerning the value assigned to the length a of the electrode. The electrode frequently enters the charge for some distance, the current in the meantime branching off into the charge. The current density will therefore gradually change in this part and the conditions assumed in our calculations will not be fulfilled. Therefore, the inside length of the electrode a should strictly be counted only to the point where the current begins to branch off into the charge. For practical purposes, however, it will be better to include also about one-half of the projecting end into the length a of the electrode.

The third assumption regarding the constancy of the conductivities of the electrode for heat and electricity is not fulfilled in practical operation, since, as well known, both increase to some extent with increasing temperature. By using mean values between the temperature limits the error which is committed will, however, be considerably reduced.

In this connection it is of interest to discuss somewhat in detail the distribution of temperature in the electrode. As already stated the heat passing through each section of the electrode will consist of the heat conducted away from the charge plus the heat generated by the current due to the resistance of the electrode between this section and the inside end of the electrode. As the distance from the inside end to the section under consideration increases, the total amount of heat passing through the section will, therefore, also increase. In order to render possible the passage of this increasing quan-

tity of heat, the temperature drop per centimeter must increase correspondingly. If the conductivities for heat and electricity of the electrode are constant, the temperature curve thus will assume the form of a parabola as shown (equation 2).

However, when these conductivities increase with increasing temperature, as is actually the case, the conditions are somewhat changed. We will still assume that no exchange of heat takes place between the electrode and the surrounding medium. If the working conditions are the same as in our first case (with constant conductivities) and the same amount of heat is conducted away from the charge, a smaller temperature drop per centimeter will now be needed at the inside end of the electrode to dispose of this heat. The amount of heat generated by the current per unit length of the electrode in this end will also be smaller than in the first case discussed, on account of the higher electric conductivity, and the increase in the temperature drop per centimeter, when gradually moving our point of observation from the inside end of the electrode, will consequently be correspondingly smaller, although growing also on account of decreasing heat conductivity. At a certain section of the electrode its conductivities for heat and electricity will correspond to the mean values assumed in the first case which we discussed.[1]

However, the amount of heat passing through this section is smaller than in the first case on account of the lower resistance of the inside part of the electrodes, and consequently the temperature drop in this section will still be smaller than in the first case. When passing further outwards the amount of heat generated by the current per centimeter of the electrode will now be larger than in the first case, due to the higher electric resistivity, while the lower heat conductivity also contributes to a more rapid drop of the temperature curve. Since the total electric resistance is the same in both cases (on account of our adoption of the mean value[2] as resistivity of the electrode in the first case) the same total amount of heat would be lost through the section at the outside end of the electrode in both cases.

In the second case the temperature drop per centimeter in this section, as well as in the part of the electrode near its outside end, will consequently be greater than in the first case. However, it is impossible to foresee, without an exact knowledge of the values of the conductivities at different temperatures, whether the temperature curves of the electrode would indicate the same temperature of the electrode in both cases at the outside of the wall. However, by defining the mean value of the heat conductivity in the first case in a special way with regard to calculations concerning electrodes, the temperatures of the electrode at the outside end will be identical in both cases. Curves *K-G-H* and *K-I-H* in Fig. 1 will give a general idea about the run of the two curves discussed.

The determination of the mean value of the heat conductivity as usually made by measuring the amount of heat conducted through an object while a constant temperature difference is maintained between its ends, will give values not suitable for our purposes.

When determining the electric resistivity of electrodes for practical purposes we measure the voltage drop and the current strength in working electrodes. The value thus obtained is not, however, the mean resistivity for the material of the electrode between the temperature limits occurring. If the latter was to be measured strictly the determination should be carried out with an electrode exposed only to a constant temperature difference, but not to a simultaneous heating by a current, using, for instance, a current too weak to influence the temperature distribution in the electrode. However, a value thus obtained would not be of practical usefulness for our purpose.

The same considerations apply when measuring heat conductivities of electrodes. In order to get values which may be directly applied in practical calculations, it will be preferable to carry out the measurements under practical conditions.

It was originally the intention of the writer to determine such values from experimental data by the aid of equation (5). For instance, an electrode (Fig. 2) may be surrounded by a highly refractory and well-insulating powder, such as magnesia. The waterjacket of the contact serves as calorimeter and the flow of water is regulated so that the temperature of the upper end of the electrode is the same as the temperature of the insulating packing at the same depth. The temperature of the electrode is also increased by means of a thermo-couple as far down as the couple will permit (1400° C.).

Under these circumstances the conditions assumed in equation (5) will hold with sufficient accuracy, and all measurements necessary for calculating from equation (5) the mean heat conductivity of the electrode may easily be made. By using such methods (properties of carbon) for determining the very high temperature of the hot end, as described by Fitz-Gerald (Trans. of Am. Electrochem. Soc., Vol. 6) and by Hansen (ib. Vol. 15) the total temperature drop in the electrode may, of course, be extended to much higher ranges.

From the nature of these methods for determining the mean

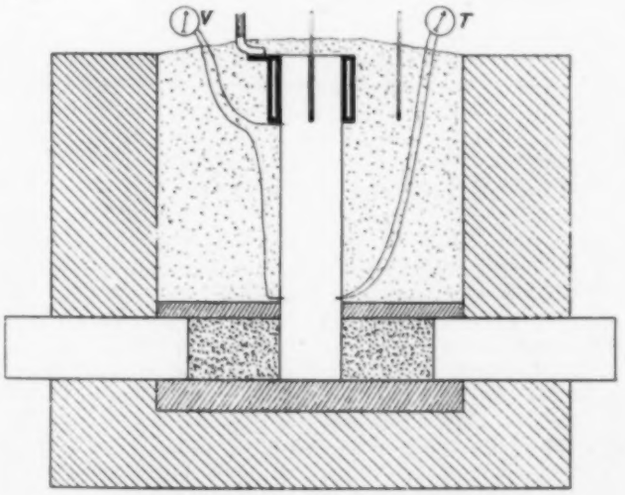


FIG. 2.—ARRANGEMENT OF TEST.

conductivities for heat and electricity, it follows that by using such values no considerable error will be introduced on account of our third assumption regarding the constancy of these conductivities.

Our fourth assumption that a uniform temperature prevails in each cross-section of the electrode will in most cases have only a small bearing on our deductions. At both ends of the electrode the isothermic surfaces will, of course, be somewhat disfigured, but for the greatest part of the electrode they will at least have a uniform shape and probably, as a rule, come close to being flat surfaces with some curving at their peripheries.

From this discussion of the assumptions involved in our deductions and the definitions made we have found that the energy loss, expressed by equation (5), will in certain cases probably correspond closely to actual conditions. The most serious objection to its correctness depends upon the assumption that no heat should be conducted away from the electrode to the surrounding wall. When this is the case the loss will be greater than indicated by the equation.

However, we may claim that the largest possible loss of energy from the charge into the electrode would occur if the heat produced by the electric current did not cause any decrease in the amount of heat which would be lost by conduction from the charge into the electrode in case no current was flowing.

[1] We assume that the general trend of the curves for these conductivities as functions of the temperature are the same.

[2] The determining of this mean value is supposed to have been made by measuring voltage drop and current strength on working electrodes.

Such a case may be imagined if the surrounding wall would cause a sufficiently strong cooling of the electrode. The total loss would then amount to:

$$W_1 = hTA/ka + ral^2/A \qquad (6)$$

As evident, the only difference between equations (5) and (6) consists in the fact that in the first-mentioned case the term ral^2/a is multiplied with the factor $\frac{1}{2}$. We will now substitute for this factor a coefficient c, the value of which may, according to the working conditions of the electrodes, vary between the limits $\frac{1}{2}$ to 1, and consequently write the equation thus:

$$W_1 = hTA/ka + cral^2/A \qquad (7)$$

where h and r express mean values, as defined, with regard to our temperature range.

This equation, therefore, represents a general solution of the problem of energy losses of electric furnace electrodes independently of any assumptions regarding heat insulation of the electrode.

Regarding the value of c we have so far only stated that it is between $\frac{1}{2}$ and 1. Its exact value in each special case will depend upon a number of considerations.

The first condition for a transfer of heat from the electrodes to the surrounding wall is that there should exist a temperature difference between them at their places of contact. In this conection we must remember that the wall is also heated by conduction from the charge.

At the inside of the furnace wall its temperature will, therefore, be about the same as that of the electrodes, no exchange of heat taking place between them at this point. On its outside the furnace wall will assume a certain temperature. Here the temperature of the electrodes will depend upon the cooling effect of the contact plates, the holder, and, possibly, the water-cooling applied. If water-cooling is used, this temperature may easily be the same as that of the wall, or close to it. In such a case there will be only a very small exchange of heat between the terminal and the wall, depending upon the difference in their temperature distribution and c will therefore have a value of practically $\frac{1}{2}$.

However, if the temperature at the outside of the terminal is considerably higher than that of the wall we must figure with a certain transfer of heat from the terminal to the wall. Another circumstance, in conjunction with the higher heat conductivity of the terminal, will offset to some extent this transfer, namely, the fact that the heat passing from the terminal to the wall will have to travel a longer path than that going directly through the terminal. Besides, there is always a contact-resistance against the transfer of heat from the terminal to the wall.

The amount of heat conducted away from the electrode to the wall will also depend upon the relation between the circumference of the electrode and its cross-section and the relation between the heat conductivities of the material of the electrode and of the surrounding wall. It is obvious that this heat quantity will be larger the larger the circumference of the electrode is as compared with its cross-section. From this fact we may conclude that in case of electrodes with uniform cross-sections, working with the same current densities and otherwise under the same conditions, the one with a larger cross-section will cause a relatively smaller energy loss and consequently give a smaller value for c.

It is also evident that the shape of the cross-section has in the same way an influence upon the energy loss incurred. Of electrodes with the same cross-sections, working under the same conditions, a round electrode will cause the smallest loss. Of rectangular electrodes a square electrode will give the most favorable results in this respect. The more the cross-section of an electrode is extended in one direction the greater will be the loss by conduction to the wall. As an extreme case we may state with great probability that if h is the heat conductivity of the electrode and h_w the heat conductivity of the wall, the value of c, derived from the following equation:

$$c = \frac{1}{2}\left(1 + \frac{h_w}{h}\right) \qquad (8)$$

will represent an upper limit of the possible value of c.

If, therefore, the relation between these conductivities is 1 to 5, as probably in the case with amorphous carbon electrodes, made from petroleum coke, when laid in brick walls, c would probably have a highest possible value of 0.60. If graphitized electrodes are used under the same conditions, the relation of the conductivities would be at least 1 to 40 and c consequently have a highest possible value of 0.51.

If the circumference of the electrode is p (A being its cross-section), the formula given for c will apply when p/A has a high value, i. e., in case of very small electrodes. On the other hand, if p/A has a small value, as is the case with large electrodes, especially of a compact cross-section, c will assume a value tending to approach $\frac{1}{2}$. For commercial furnaces the value of c will therefore probably be located closer to $\frac{1}{2}$ than to the value found from equation (8).

Although in this way c is shown to be a function of two variables, i. e., the cross-section and circumference of the electrode, the possible variation of c appears to be comparatively small. Instead of attempting to introduce into our calculations an expression for c (for instance, $c = \frac{1}{2}[1 + h_w p/(hp + hfA)]$ where f is a constant) so as to take into account the considerations presented, thus complicating our deductions and conclusions in a high degree, we will therefore attribute a constant value to c, depending upon the type of the furnace and, within certain limits, upon the size of the electrode.

In order to give equation (7) a form more suitable for our deductions we substitute for A the expression: $A = I/d$ and thus obtain:

$$W_1 = I\,(hT/(kad) + crad) \qquad (9)$$

It should be noted that this energy loss is expressed in watts. As the current I is one factor in this expression it follows that the other factor represents the *total voltage loss* in the inside part of the electrode, due to the combined loss of energy by heat conduction from the charge and on account of the resistance of the electrodes.

As the total loss of energy W_1 consists of the sum of the loss on account of the resistance of the electrode $radI$ and the loss by heat conduction from the charge Q (i. e., the heat conducted from the charge into the electrode) we may get an expression for the last mentioned loss by means of equation (9):

$$Q = I\,[hT/(kad) - (1 - c)\,rad] \qquad (10)$$

Evidently, by using a certain current density, the conduction of heat Q from the charge into the electrode may be entirely avoided. This is the case if:

$$d = \frac{1}{a}\sqrt{\frac{hT}{kr\,(1 - c)}} \qquad (11)$$

The electrode loss, when no heat is conducted away from the charge, amounts to:

$$W_1 = I\sqrt{hrT/k\,(1 - c)} \qquad (12)$$

As the electrode loss in this case is caused only by its electric resistance, we may also express this loss in the following way:

$$W_1 = radI \qquad (13)$$

This expression may, of course, also be deduced from a combination of equations (11) and (12).

From equation (9) we also conclude that the total electrode loss will have the smallest possible value for a certain value of the current density. This value is:

$$d = \frac{1}{a}\sqrt{\frac{Th}{ckr}} \qquad (14)$$

The smallest possible loss, therefore, amounts to:

$$W_1 = 2I\sqrt{Tchr/k} \qquad (15)$$

We have considered so far only electrodes surrounded by an insulating wall. However, it is well known that electrodes may

enter the furnace also in other ways. Thus they may be suspended vertically in the furnace and the charge fed down around them, as in the case, for instance, in carbide or ferrosilicon furnaces. In other cases, for instance, in the steel furnaces of Héroult and Girod, the electrodes enter the furnaces through openings in an arched roof and do not come into direct contact with the charge.

In the first-mentioned case our formula may usually be applied. The surrounding charge will, as a rule, conduct more or less electric current. Its resistance, however, will be considerably higher than that of the electrode and, besides, the contact resistance between the terminal and the charge will prevent to a great extent a spreading of the current into the upper layers of the charge. The electric conductivity of the charge will, therefore, as a rule, not constitute any serious objection to the use of our formula.

Regarding the question of heat losses from the terminal to the surrounding charge we must consider the fact that any heat possibly given off from the terminal to the charge will preheat the charge and, therefore, cannot be charged as a loss.[1] At the same time it is of greatest importance that no heat should be conducted away by the terminal from the zone of highest temperature at its lowest end. This will depend upon the temperature distribution in the electrode.

At the lowest end of the electrode, where reduction and melting of the charge takes place, highly heated gases are formed which pass upwards through the charge around the terminal. These gases are therefore cooled and their heat is absorbed by the charge. In the lower part of the charge, near the end of the terminal, heat is also supplied to a great extent by conduction from the electrically heated zone. The temperature drop per centimeter (counted upwards) will therefore be small in this part. Higher up, where the heat is mostly carried by the gases, the drop will be faster.

It will be remembered from our discussion that in the first case considered the temperature in the electrode varied according to a parabola and that when no heat was lost from the charge by conduction through the electrode, this parabola had its center at the inside end of the electrode. This distribution of the temperature in the terminal, therefore, agrees in its general features with the temperature distribution in the charge as described.

There seems, therefore, to exist no principal objection against the application of our deductions to electrodes in furnaces in which the charge is fed down around the electrodes, due consideration being given in each case to the circumstances which influence the value of c. The drop in temperature T in our calculations represents in this case the temperature difference between the hottest end of the electrodes and the part of the electrodes at the top of the charge and a the distance between these points.

In the third class of furnaces, with electrodes freely suspended, the conditions are much more complicated. Our equations, however, undoubtedly may be applied also in this case. An account of the probable conditions prevailing in these electrodes, however, would involve so many uncertain assumptions that a detailed discussion of this subject may preferably be postponed until experimental data are available regarding the questions involved.

We have so far considered only the losses of the part of the electrodes inside the furnace. The protruding end of the electrodes will, of course, cause an additional certain loss due to its resistance. This loss may be expressed in watts by the following formula:

$$W_2 = rldI \qquad (16)$$

The electric resistivity r in this case has a higher value than obtains in the inside part of the electrode due to the lower temperature of the outside part.

[1] As our expressions for W_1 include also the heat, conducted away by the surrounding medium, these expressions will therefore give somewhat too high values (if $c > \frac{1}{3}$) in case of furnaces of this type.

As the outside end of the electrode should, from every consideration, be made as short as possible, we will not further enter upon a discussion of equation (16), but limit our deductions to the inside part of the electrode.

We may draw a number of conclusions of practical value from the equations derived. First, however, a general remark deserves attention. We have found that the heat conductivity of the electrode is of exactly the same importance to the working of the electrode as its electric resistivity. The electric resistivity has until recently been practically the only property which has been taken into consideration, both by furnace men and by manufacturers, especially when favorable as a "talking point." With electrodes of different materials, the loss is proportional to the square root of the product of the thermal conductivity h, and the electric resistivity r, other things being equal. If in case of two electrodes, A and B, made from different materials, the electric resistivity of A is only one-fourth of that of B, but the heat conductivity of A is eight times higher than that of B, then the smallest possible loss of energy in A would be 41 per cent higher than the smallest possible loss in B in spite of the lower electric resistivity of A. From the point of view of energy losses incurred, a low electric resistivity is therefore no inducement to the use of an electrode if its heat conductivity is very high.

In discussing the energy losses of electrodes the greatest interest, of course, centers around the equations expressing the most favorable conditions attainable with regard to these losses. Equations (11) to (15) refer to these conditions. It is obvious from these equations that when c possesses the value $\frac{1}{3}$, as probably is often the case, at least approximately the values of the current density d, according to equations (11) and (14), will be identical. In such a case, therefore, the same current density will give the smallest possible electrode loss and, at the same time, prevent any loss of heat from the charge by conduction through the electrode.

If c has a higher value than $\frac{1}{3}$, the electrode loss, equation (14), will be a minimum for a current density with which a certain amount of heat flows from the charge into the electrode. In order to avoid this loss it would, therefore, be necessary to raise the current density to a higher value, according to equation (11). If we assume that c has a value as high as 0.75 this raise of the current density would amount to 73 per cent, while the corresponding increase in the total loss of energy in the electrode would only amount to 15 per cent. If c, however, possesses a more normal value, such as 0.6, the corresponding percentages only amount to 22 per cent and 2 per cent. These figures for the increase of the energy losses, when compared with other losses in the furnace, are so insignificant and the advantages gained from these small sacrifices of energy so important that we may state it as a general rule that if other considerations do not interfere, the lowest current density, as given by equation (11), should be used which will cause no loss of heat by conduction from the charge into the electrode.

This loss will also be prevented if a still higher current density is used, but the total loss of energy will, in such a case, be further increased, as may be seen from equation (9). This case is interesting because it corresponds to certain results obtained by Mr. Hansen. He found in an experiment that the total loss of energy of his electrode was less than corresponded to the Joulean heat of the electrode as figured from its resistance.

It is obvious that the necessary requisite for avoiding any loss of heat by conduction from the charge is that in some section of the electrode there should be no temperature drop (per centimeter). For instance, in case of the electrode in Fig. 1, the tangent of the temperature curve at some point should be horizontal. To accomplish this it is only necessary to use a current density which will produce this turning point of the temperature curve right at the inside end of the electrode; the temperature is then at this end the same as prevails in the furnace (Fig. 1, curve *F-M-H*). This current

density has been defined by our equation (11). If a higher current density is used, the top point (L) of the temperature curve F-L-H will fall within the wall and the heat generated from electrical energy will flow both ways from L, part of it being lost in the ordinary way, the other part being supplied to the furnace. This consequence of our deductions has been realized in the case of Mr. Hansen's experiment. Although Mr. Hansen was able in this way to get a smaller loss than corresponding to the resistance of his electrode, he nevertheless had a larger loss than necessary. With a somewhat lower current density, it would have been possible to avoid any heat conduction from the charge into the electrode with a smaller energy loss in the electrode.

There is another side to this question. If such a high current density is used that an overheating of the electrode occurs within the wall, the temperature attained will often ruin the wall, with consequent disastrous results to the furnace, certainly a sad experience of many electric furnace workers. On the other hand, if the same condition prevails in a furnace where the charge is fed down around a vertical electrode, the overheating of the electrode may cause a clogging of the charge around it and a wasteful leakage of the current. If in such furnaces two electrodes of different polarity are suspended side by side, the undue conductivity thus imparted to the charge between the electrodes will sometimes cause a short circuit and the current will not pass through the molten bath below the electrodes, as intended.

If we have electrodes with a thermal conductivity h and electric resistivity r, and want to use them in a furnace with a total temperature drop in the electrode T, and a current I, we may assume for practical purposes that c will have a certain fixed value independently of the length a we may give the electrode. Equations (11) and (12) may then be written as follows:

$$d = C_1 \sqrt{T/a} \qquad (17)$$

$$W_1 = C_2 I \sqrt{T} \qquad (18)$$

where C_1 and C_2 are two constants.

The product $C_2 \sqrt{T}$ in equation (18) represents the total voltage loss in the electrode when no heat is lost by conduction from the charge, the current density required for this purpose being given by equation (17). This current density is inversely proportional to the length of the electrode, while the voltage loss thus incurred is independent of the length of the electrode, remaining a constant for a certain total temperature drop in the electrode. A practical application of this rule is that if a certain current density has proved to give the most satisfactory results obtainable in a furnace, we may in this way compute the best value of the current density for another furnace of the same kind but with different length of the electrode. Because this principle has not been recognized there has certainly resulted much distress and surprise on the part of electric furnace men, the troubles being mostly due to the overheating of electrodes, already discussed.

With a certain total temperature drop in the electrode it is, therefore, feasible to incur the smallest possible loss of energy in the electrode whatever the length of the electrode may be, if only the current density is accordingly adjusted. In order to reduce the cost of electrodes, therefore, it would seem rational to use very short electrodes with a correspondingly high current density and to prevent an excessive heating of the outside end of the electrode by water-cooling. Although this method undoubtedly may be used with advantage to a larger extent than practiced so far, its application in most cases is limited by practical considerations.

In furnaces with fixed electrodes, surrounded by a wall, the method may sometimes prove useful if the water-cooled metallic holder is allowed to enter the wall for some distance. However, the risk of burning off the electrode, with all its dangerous complications and the difficulty in making good contact on the inside between the electrode and the charge or perhaps the permanent resistor will limit our endeavors in this direction. Besides, as the cost of the permanent electrodes in this class of furnaces is a comparatively small item, this possibility of using very small electrodes has no great bearing on electric furnace construction.

On the other hand, in many cases, especially with furnaces where the charge is fed down around the electrode, the best current density is dictated not only by electrode considerations, but also by the conditions at the place of transfer of the current from the electrode to the charge, where, as a rule, most of the heat in the furnace is generated. The electrode—its dimensions as well as its quality—should therefore be adjusted largely according to these conditions.

If a current density too high for the purpose of the process is used, the heat generated here may be too excessive to be usefully disposed of. Parts of the charge may be needlessly volatilized, products, already formed, may be decomposed, and undesirable reactions may take place. Besides, in such a case the electrode itself will be attacked and consumed to a much greater extent than necessary. It is probably a fact that from considerations to the first cost of electrodes a higher current density is used in many cases than consistent with the lowest consumption of electrodes per ton of output.

If we have furnaces of the same kind, but with different temperature drops T in the electrodes, the factors C_1 and C_2 assumed as constants in equations (17) and (18) will no longer remain so. The mean conductivity for heat h and the electric resistivity r will assume different values in the different cases. However, if there is no considerable difference in the total temperature drops we may assume that the relative increase in the mean thermal conductivity will be approximately the same as the relative decrease in resistivity and that, therefore, the agreement of their product will remain practically a constant. Under the same assumption there cannot exist any considerable difference between the values of c. Even if its different values varied as much as, for instance, from 0.55 to 0.60 the increase in the value of $\sqrt{1/(1-c)}$ would only amount to 6 per cent. We therefore find that the value of C_2 in equation (18) may be regarded as a constant for practical purposes when there are no great variations in the total temperature drop. However, the value of C_1, containing as a factor the square root of the quotient of h and r, will be influenced by a change in the total temperature drop to a degree which we cannot determine in advance. The only rule we can give at present for the most suitable current density in case of an increase of the total temperature drop in an electrode of a certain length a is therefore, that this current density should be higher than the one obtained by multiplying the best current density at a lower temperature drop by the square root of the quotient of the larger and the smaller temperature drops.

For the reasons given we may consider that equation (18) will hold with sufficient accuracy also in case of different temperature drops in the electrode within certain limits. As a rule, the inside temperature of a furnace used for a certain purpose is determined by the process worked. A comparison of the losses in the electrodes in case of different processes is therefore not of the same interest as an analysis of the possibilities offered to reduce the loss in a certain case. We have already at length discussed this subject, assuming a constant temperature drop. However, different temperature drops may occur in electrodes, having a definite temperature at their inside ends, on account of different temperatures being maintained at their outside end. Equation (18) may be safely applied to such cases. We may derive some important conclusions regarding the effect of water-cooling the outside end of the electrode upon the loss incurred.

The outside end of an electrode, exposed to the air, must not assume a temperature causing a burning of the electrode. If the protruding end would become red hot without water-cooling, its application is imperative. If the current density used corresponds to the value defined by our equation (11) or

(17), water-cooling in such a case cannot be regarded as causing an additional loss. It would be possible by giving the electrode a greater length and using a lower current density according to equation (17) to produce the same temperature at the outside end without water-cooling, but the loss, incurred in the electrode, would nevertheless be the same in both cases, as shown in our previous deductions.

A temperature drop reaching down to the burning temperature of carbon is unavoidable in any case, and we should, therefore, only charge the loss incurred on account of the additional drop in temperature against the water-cooling applied. This additional drop will, at the most, amount to about 400° C. If the inside temperature of a furnace is 2000° C., and the temperature drops in the electrodes are 1950° C. with water-cooling and 1550° C. without water-cooling, then the additional loss, according to the equation (18), on account of the water-cooling, would amount to 12 per cent of the loss without such cooling. If the voltage loss, due to the electrode, is 1 volt without cooling, then the additional loss, caused by its application in this case, would only reach 0.12 volt. Besides, the higher the inside temperature of the furnace is the smaller will be the relative increase of the loss due to the water-cooling.

We therefore find that water-cooling, when applied to electrodes working with proper current densities, does not cause any significant losses. Those incurred are more than compensated for by the benefits.

It may be appropriate in this connection to discuss somewhat the practical advantage of water-cooling, as this subject is closely related to the conditions for the best working of electrodes. The purpose of the water-cooling is not limited to the purpose of preventing the burning of the electrode. Water-cooling is also of the greatest importance for the maintaining of a good contact between the cable connection and the electrode. As well known, such a contact is usually accomplished by firmly pressing metallic contact plates against the electrode. When possible (compare the article by FitzGerald in Vol. 3, 1905, page 14, of this journal) this contact is often improved by placing a layer of graphite between the plates and the electrode. In many cases, however, the application of such a layer is impracticable and the contact resistance produced may therefore be unduly high on account of the combined influence of a high specific contact resistance and of the fact that the plates and the electrode actually touch each other only at a few spots.

Without water-cooling a very strong heating of the contact place may follow, resulting frequently in oxidation of the plates. This increases the contact resistance still more with the consequences of a burning of the electrode and arcing contact. Water-cooling will largely prevent these troubles and therefore serve to keep the electrodes in running order. However, if the contacts have not been carefully made, water-cooling will conceal the bad condition of the contacts and the waste of energy on their account will be, as a rule, attributed to the electrode alone, if recognized at all. This serious cause of energy losses deserves more attention than is usually bestowed upon it.

As an example of the losses sometimes occurring on account of bad contacts, an occurrence within the writer's experience may be cited. The connection to a permanent electrode carrying 1800 amp was made by firmly clamping against two large copper plates with interposed layers of graphite ("decomposed carborundum"), the current density in the contact surfaces being only 11.4 amp per square inch. When the connection was first made, a voltmeter, permitting readings of 1/200 volt, only indicated a voltage drop of about 1.07 volt in the connection. After three runs of the furnace the electrode became excessively hot and on renewed reading of the voltage drop it was found to have risen to 1.47 volt. The voltage drop in the electrode itself amounted to 1.16 volt. This enormous energy loss in the contact was found to be due to a tarnishing of the copper plates from fumes emerging from the furnace. This instance may serve as a caution against similar occurrences of wasteful losses.

A short outline of a simple way to calculate the dimensions of electrodes, based upon the above formulas, may now be of interest.

In a given case the inside temperature of the furnace may be regarded as known. Although this is one of the fundamental quantities for any calculation of this kind, its knowledge is far from satisfactory in most cases on account of our limited knowledge of the temperatures required for the reactions or processes contemplated. Sometimes we may find from preliminary experiments some guide for estimating the temperature required. An examination of the electrode may give important clews, as shown by Mr. Hansen. However, the results obtained in this way must be interpreted with great caution if the experimenter has not an exact knowledge of the quality and composition of the electrode used. Otherwise an extensive investigation of the same kind as carried out by Mr. Hansen will be necessary.

However, we may find other indications of the temperature attained. The carborundum frequently formed on the tips of the electrodes may render such assistance. From the measurements of Tucker and Lampen (compare the synopsis in this journal, Vol. 4, 1906, page 363) we may conclude that the temperature in a given case has been between 1615° C. and 1950° C. if green amorphous carborundum is found on the electrode, but no crystalline carborundum. If crystalline carborundum has been formed, but no crystals of "decomposed carborundum" (graphite) are found, the temperature has been above 1950° C. and probably not exceeded 2220° C., but the graphitizing of the electrode tip does not necessarily indicate a temperature above 2220° C. Some other valuable indications may also be obtained from data given in the original paper of Tucker and Lampen (*Jour. Amer. Chem. Soc.*, July, 1906). It is to be hoped that an extended knowledge of the "chemistry of high temperatures" as well as of the physics of those temperatures will soon give us an exact basis for work of this kind.

Generally electric furnace workers have a tendency to exaggerate the temperatures prevailing in their furnaces. It is a common claim that the charge has been heated to the temperature of the arc, citing 3500° C. to 3700° C. as this temperature, because an arc was drawn in the furnace. The experimenter forgets that said temperature prevails when the arc is extended between electrodes of pure carbon with carbon vapor alone as conductor and not in a furnace usually filled with metallic vapors of comparatively high pressures. Further, the constituents of the charge and the products formed, as a rule, have a high vapor pressure at these temperatures, the sum of which may reach atmospheric pressure, preventing an increase of the temperature on acount of the subliming or boiling which then ensues.

Magnesia, one of our very best refractories, melts at 1920° C. (Goodwin and Maily, this journal, Vol. 4, page 217) and boils at a temperature shortly above its melting point according to Moissan, which statement is confirmed by observations of the author. In most extreme smelting processes we will probably not be far from the truth in assuming that the temperature will be somewhere between 1800° C. and 2200° C. When heating solid charges, such as certain carbides or carbon, the temperature may, of course, run up considerably higher.

When the inside temperature has been determined or estimated in one way or the other, the outside temperature must be decided upon. If water-cooling is used no appreciable error will be committed in placing this temperature at 100° C. Otherwise it may be about 100° C. or 200° C. higher.

The length of the electrode is the next consideration. This dimension will depend upon the working conditions of the furnace, and no general rule can be made for its proportioning except that the length is limited by the technical difficulties in manufacturing (preventing at present the manufacture of amorphous carbon electrodes of greater length than 7 ft.).

When the temperature drop T and the length a are thus fixed, the current density d will be the important object of our attention.

As already stated, the best current density is determined not only by the electrode, but also to a great extent by the purpose of the process worked. With regard to the last mentioned consideration we will usually have some latitude for fixing the current density, and we may therefore first investigate what density is most suitable on account of electrodes of different materials available. In order to do so we have to know the various physical constants of the electrodes and the electrode constant c of the furnace.

With a knowledge of the progressive and scientific spirit of the concerns manufacturing electrodes in the United States, we may confidently hope that they will carry out such investigations as will place to the disposal of their customers the information wanted regarding their electrodes.

For furnaces with an electrode constant c, which has the value $\frac{1}{2}$, we may calculate from equation (12) the voltage drop $\sqrt{hrT/k\,(1-c)}$, which corresponds to the temperature drops T when no heat is conducted away from the charge. Experiments should then be carried out with the same kind of furnace and using the calculated current densities to check whether the energy actually lost corresponds satisfactorily to the quantity calculated from equation (13). If this is the case a very nice check has been obtained for the correctness of the determinations made.

The next step in the investigation required is to determine the energy losses in the electrodes under working conditions of furnaces in order to find the value of c under different conditions.

Such experiments will surely offer certain difficulties, which, however, may be overcome. When the value of c is known in a given case, we may find the smallest voltage drop required in the electrode in order to avoid heat conduction from the charge by multiplying the voltage drop for $c=\frac{1}{2}$ with the correction factor.

Having access to the results of these determinations, the designer may draw a curve showing the smallest voltage drops in the electrode at different total temperature drops, when no heat is conducted away from the charge, assuming the value ½ for c equation (12).

In a given case in practice, characterized by a certain total temperature drop and a special value of c, he will take from the curve the voltage drop, corresponding to the temperature drop in question, and multiply this voltage drop by the correction factor $\sqrt{\frac{1}{2}\,(1-c)}$. He will find in this way the voltage drop which should prevail in his electrode, regardless of its length, in order to avoid heat-conduction from the charge into the electrode.

By resorting now further to equation (13), he finds that this voltage drop should equal rad, and since the mean electric resistivity r of the electrode at this total temperature drop is supposed to be known and the length a has been decided upon, the most suitable current density d, with regard to the electrode, is found by a simple division.

It is possible, and even probable, that this current density does not correspond to the value, desirable on account of the process to be carried out. In such a case, of course, the original assumed working conditions of the electrode may be adjusted in order to satisfy both considerations, or some other compromise may be effected. It is also perhaps possible to select electrodes of another and more suitable material in such a case.

In one way or the other way a set of conditions may thus be worked out from calculations supported by practical experiments, which will give ultimately the best economical efficiency of the furnace electrodes with regard to the electrode losses and costs and to the process worked.

HÖGANÄS, SWEDEN, September, 1909.

Cyanidation of Manganese-Silver Ores.

Some of the metallurgical difficulties which have been encountered in the cyanidation of manganese-silver ores in Mexico are brought out in two communications by Mr. ROBERT LINTON (of Los Angeles, Cal.) and Mr. E. M. HAMILTON (the well-known metallurgist of the Butters interests) to the *Journal of the Chemical, Metallurgical and Mining Society of South Africa*. They are published in the issues of March and August, 1909, respectively.

The problem with this special class ore is to unlock the silver in the manganiferous ore so as to make it soluble in potassium cyanide solutions. The insoluble silver is contained principally in the highly oxidized surface ore between the croppings and the 200 level, while ore from the bottom of the mine usually yields 90 to 95 per cent of the silver without any trouble.

Mr. ROBERT LINTON found a solution of the problem in preliminary chloridizing the ore. By treating with potassium chlorate and hydrochloric acid, or roasting with salt and sulphur, and following this treatment by weak KCy solutions in the usual manner, a uniformly high rate of extraction was obtained.

For operation on a practical scale Mr. Linton proposes to apply the common laboratory method of preparing chlorine, using salt, manganese dioxide and sulphuric acid according to the reaction:

$$2NaCl + MnO_2 + 3H_2SO_4 = MnSO_4 + 2HNaSO_4 + 2H_2O + Cl_2.$$

The salt was mixed with the ore, crushed through 100-mesh screen. The sulphuric acid was added in weak solution (5% H_2SO_4) and allowed to stand with occasional agitation for 24 hours. The solution was then filtered off, the residues washed and treated with 0.2 per cent KCy solution. The silver extraction varied from 60 per cent to 94 per cent. The method has the advantage of low cost of materials; commercial sulphuric acid and salt are cheap, and the manganese dioxide is already present in the ore. So far the method has been applied by Mr. Linton on a laboratory scale only.

Mr. E. M. HAMILTON, in his quite extended experiments with manganese-silver ore, has reached somewhat different conclusions. He confirms the difficulty of dissolving the silver. He tried cyanidation with all strengths of cyanide, even up to 5 per cent, on samples of the ore ground to pass 200 screen, and could only extract from 5 per cent to 15 per cent of the silver. He tried the addition of all kinds of reagents, such as lead salts, oxidizers, chlorides, oxygenation with air, hot solutions, etc., without effect.

Since the difficult ore is from the highly oxidized rock, while the ore from the bottom of the mine offered no difficulties, Mr. Hamilton concluded to treat the difficult ore first with a reducing agent. He accordingly subjected the finely ground ore to treatment with solutions of various reducing agents, among which were hydrogen sulphide, sodium sulphide, sodium hydrosulphide, and roasting with sulphur in a closed vessel. The charges were agitated for about 24 hours, and then well washed with water and cyanided by agitation in bottles.

The sodium sulphide had no effect whatever, but both the ammonium and sodium hydro-sulphides and the hydrogen sulphide increased the subsequent silver extraction to about 73 per cent against about 5 per cent by straight cyaniding. The cyanide consumption was also increased from about ½ lb. per ton in straight cyaniding to 12 lb. per ton for the samples treated by hydro-sulphide and to 27 lb. for that treated by hydrogen sulphide. The resultant cyanide solutions carried large quantities of thiocyanate.

The samples roasted with sulphur have about the same yield as those treated with hydro-sulphide, and about the same cyanide consumption.

To try to reduce this heavy loss of cyanide, the ore, after reduction treatment, was given various treatments before cyanidization, among which were the application of chlorine water and various chlorides and of oxidizers, such as potassium permanganate and hydrogen peroxide, and also the blowing of air through the

pulp; they were then well washed before cyanidation. In the case of intermediate oxidation tests some cyanide was saved in the subsequent treatment, but the extraction of silver was considerably less than when the ore was cyanided immediately after the reduction treatment. The intermediate chlorine treatment made no difference in the cyanide consumption, but increased the silver extraction by 3 per cent or 4 per cent.

Owing to the excessive cyanide consumption and the low value of the ore ($8.00 to $10.00 per ton) this method of treatment by reducing agents could not be entertained, although if the cyanide loss had been normal the method would have been well worth considering. The reason why this treatment increased the silver extraction was rather mysterious, and attempts were then made to ascertain the mode of occurrence of the silver.

By grinding the ore to pass 200 screen and boiling for an hour in strong nitric acid, then washing well with distilled water and assaying the residue, the latter was still found to contain 75 per cent of the original silver content. This residual silver insoluble in nitric acid could not have been in the metallic or sulphuric forms, nor was it mechanically inaccessible in the ordinary sense, because a sample of the pure clay slime from the ore was even more refractory to nitric acid than the general sample. It might have been in haloid form, but in that event it should have been soluble in cyanide or sodium thiosulphate, which reagents had practically no effect on it.

About this stage of the inquiry Mr. Hamilton obtained the assistance of Mr. G. H. Clevenger, who made many experiments, of which the following is a short summary: (1) Bromocyanide, results rather inferior to those by straight treatment. (2) Addition of zinc dust as a reducing agent, no result. (3) Addition of metallic sulphides with a view to breaking up any hypothetical haloid compounds, no result. (4) Addition of strong alkali with and preliminary to cyanide treatment, no result. (5) Preliminary treatment by pan amalgamation, no result. (6) Preliminary treatment with chlorine and bromine prior to cyaniding, no result. (7) Preliminary roast before cyaniding, result—not a trace of silver soluble in cyanide. (8) Preliminary chloridizing roast, result—fair silver extraction (75%). (9) Concentration of the ore ground to 200 mesh, result—tailing assayed rather higher than concentrate. (10) Oil concentration, no result. (11) Acid flotation, no result. (12) Preliminary treatment with hydrosulphides, results confirmed the tests made by Mr. Hamilton as described above. (13) Preliminary treatment with hydrochloric acid, result—94 per cent extraction of the silver.

From the foregoing it appeared that the silver is in very intimate association with some other mineral, probably with the manganese, possibly occurring as a definite manganese-silver compound.

As to how the silver may be made amenable to cyanide solution, however, Mr. Hamilton concludes that chloridizing is not the only, and probably not even the best, method. He is led to think that instead of trying to attack the silver, which is comparatively small in quantity, the aim should be to attack the predominant partner in the combination—that is, the manganese.

In the above-detailed tests a chloridizing roast yielded only 75 per cent of the silver on subsequent cyanidation, while the preliminary treatment with hydrochloric acid allowed an extraction of 94 per cent. Of course, in this latter instance it may be assumed that the result was due to nascent chlorine formed by the manganese dioxide; it might, however, be equally well merely due to the dissolving action on the manganese, exposing the contained silver to the action of the subsequent solvent, and this view would seem to be strengthened by the fact of the chloridizing roast giving results so much inferior to those of hydrochloric acid.

But if this latter view be correct, how are we to account for the results due to a preliminary reduction with hydro-sulphide? The only suggestion Mr. Hamilton has to offer is that the manganese, being converted into sulphide, became sufficiently soluble in the cyanide solution to expose most of the silver to the action of the solvent, forming thiocyanate and mangano-cyanide of potassium.

Working on the hypothesis of the necessity of finding some solvent that would break up the combination and free the silver, he tried a preliminary treatment with many reagents. (1) Nitric acid had been found useless in the early stages of the investigation, probably because it did not attack the manganese dioxide. (2) Hydrochloric acid, while very effectual, appeared to be prohibitive in cost. (3) Sulphuric acid was without effect. (4) The hydro-sulphide treatment, while being effectual, was prohibitive on account of the loss of cyanide, though he found it possible to regenerate KCN from KCNS by electrolysis with caustic soda, and it might have been possible to recover some of the cyanide in this way. (5) Finally he tried 5 per cent solution of sulphurous acid. After washing and cyaniding the residue he obtained an extraction of 84 per cent of the silver with a cyanide consumption of only 4 lb. per ton of ore. This method seemed by far the most feasible of any, and if the ore had assayed a few dollars higher than it did, the method would certainly have been tested on a working scale. The amount of sulphur required would in this instance have been about 100 lb. per ton of ore, which was more expense than the grade of rock would have warranted.

Thus Mr. Hamilton concludes from the foregoing results that there are several methods of treatment besides chloridising which were found to give good cyanidizing extractions on the silver in this manganiferous ore: (a) Preliminary roasting with sulphur in absence of air. (b) Preliminary treatment with alkaline sulphides. (c) Preliminary treatment with hydrochloric acid (which may be considered to fall under the chloridizing class). (d) Preliminary treatment with sulphurous acid.

Of all these the last seems to Mr. Hamilton the most practicable and least expensive, and with an ore assaying a little higher than that which was experimented on, would seem to offer a workable solution of the difficulty in the cyanidation of these manganese-silver ores. It may be remarked that the gold in such ores usually yields a uniformly high extraction of about 90 per cent under almost any circumstances, regardless of the behaviour of the silver.

Why Is Electro-Steel Sometimes Unhomogeneous?

By Assar Grönwall.

Several steel manufacturers in England, Germany, and the United States have found that electrically produced steel is at times unhomogeneous. In one and the same ingot the one end may be of high quality, while the other end is quite inferior. This complaint has been made too often to accept the usual explanation that in the beginning of electric steel manufacture part of the steel which was placed on the market was not yet up to perfection, and that ever since the conservative crucible-steel manufacturers, unwilling to abandon their crucible furnaces, emphasized on all occasions the lack of homogeneity of electric steel.

The main part of the high-class steel hitherto produced in the electric furnace has been made from cold raw materials, viz., pig iron and scrap. The furnaces used have principally been of the following three types: First, single-phase electrode furnace with two adjustable electrodes (Héroult). Second, single-phase electrode furnace with one adjustable electrode and one bottom electrode (Girod). Third, induction furnace.

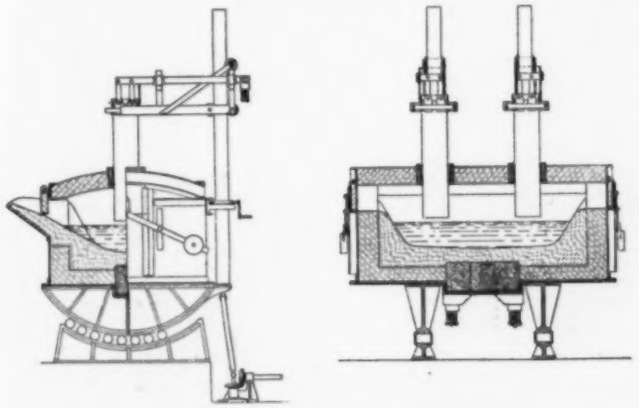
An investigation of the principles of operation of the different systems will give an explanation why lack of homogeneity is sometimes found in the steel. As to the Héroult furnace, after the main part of the charge has been molten the current passes from one electrode through an arc to the slag, then through the slag and the upper part of the metal, and through another arc back to the second electrode. The main part of the heat is produced in the two arcs, a smaller part in the slag, and only a very small part in the metal. Thus the metal is heated

through conduction from the slag, and may therefore not be uniformly heated; the surface is always hotter than the metal near the bottom, as the hotter metal is lighter than the cooler.

Therefore it may happen that if, for instance, a large piece of scrap has been thrown in the furnace with or after the other charge, or if such a piece has come to be lodged in a corner of the smelting chamber, this piece may be barely molten when the charge seems to be ready for tapping, to judge by the look of the slag or from the tests. If, then, the metal is tapped, the hottest and purest steel will run out at first, and the cooler unrefined metal at last.

In steel furnaces with one adjustable electrode and a bottom contact, just the same can happen; however, this type is in this respect more advantageous, since some part of the heat here is developed in the bottom.

In the induction furnace the same conditions may also exist. In it the electrical energy is converted into heat in the bath itself. If the lining is properly made, the cross-section of the bath is the same all over the annular hearth, and the development of heat is uniformly distributed. But if, for instance, after some time the lining in some place is eaten away and thus the cross-section enlarged, less heat is developed just at that



SWEDISH ELECTRIC STEEL FURNACE.

place where rather more heat is wanted, since more metal is accumulated there. This unequal temperature at different parts of the bath may, of course, lead to non-homogeneity of the steel.

A new type of electric steel furnace recently patented and placed on the market by the Swedish Elektrometall Company seems to fulfill better the conditions for getting a homogeneous steel. This furnace is shown in the adjoining illustration. The electrical energy is supplied in form of two-phase currents, one of the terminals of each of the two phases being connected to the two adjustable electrodes, while the two other terminals are both connected to the bottom contact. The two-phase currents produce a rotating magnetic field in the bath, which causes a circulation of the metal sufficient to get a homogeneous product and a uniform temperature in all parts of the bath. Since the circulation takes place here in a vertical plane, the metal near the bottom is continuously brought up to the surface and the hot slag.

In all the above types of furnace it is important, when producing high-class steel, that the time for treating one charge is sufficiently long. A prolonged treatment also makes sure that the metal is completely freed from gases and slag. This is of great importance for the quality of the steel, although it is as yet impossible to prove the presence of gases by analysis.

It is, of course, necessary to tap all the steel into a ladle before tapping into the moulds in order to get the metal thoroughly mixed. The author has seen at one steel works that the steel was tapped from the furnace (an induction furnace) into a small ladle, taking only a quarter of the charge, so as to run metal into the moulds, placed on a rotating table, simultaneously with the tapping from the furnace. Since the metal could not possibly be well mixed, this procedure is not recommendable.

Finally, it is to be added that the production of high-class steel always requires the greatest accuracy as well as long experience, and the production of electric high-class steel is by no means an exception from this rule. The attendance of an open-hearth furnace is never left to a person with less than one year's experience, and it is not to be expected that it is otherwise when dealing with an electric furnace. The electric furnace has been in practical use for too short a time. There are to-day not yet many men thoroughly familiar with the running of an electric steel furnace for high-class steel. But after some years' experience the electric furnace will displace the crucible furnaces, which are too uneconomical.

Calculations for Dryer Design*

BY DR. WILLIAM M. GROSVENOR.

The following methods were developed as a matter of commercial necessity. They were needed to design a great number of drying installations adapted to meet widely different requirements, operating under a system the advantages of which have hitherto been largely overlooked, i. e., the repeated circulation of the same air over condensing coils to remove moisture, over heating pipes to increase the "moisture capacity," over the material being dried and again over the condenser.

Surprising steam economies are thus accomplished, in some cases amounting to more than 80 per cent. Insurance risks are reduced. So-called "moist drying" becomes a matter of course, great capacity is permitted, and complete independence of climatic conditions makes accurate design and perfect regulation possible.

The engineering of drying is to-day one of the most neglected and by no means one of the most simple branches of applied science. There are thousands of installations where the air travels 10 feet in idleness to one of useful occupation, where it is taken from the streets and the gutters and blown loaded with bacteria over fermentive products (many of them food stuffs), where it is received ice-cold and heated up 50° or 100° before it begins to do its profitable work, or is drawn in nearly saturated at 85° to dry a product that cannot safely be heated above 95°, where, worst of all, it is expected to reverse the performance of the widow's oil jug, and to keep on drinking up moisture indefinitely without renewal or dehydration. Taking advantage of the variable weather conditions, which the seller of machinery had no object in controlling, the machinery was made big enough—the bigger the better.

In developing simple, rapid, and at the same time reasonably accurate methods of design, we were met at once with the fundamental difficulty that there is no simple equation expressing the relation between the tension of aqueous vapor and the amount of water, either by volume or weight, saturating air under given conditions.

The next difficulty was that the base or divisor (cubic foot) of all humidity calculations has hitherto been most unfortunately selected, as it varies with every change of pressure, temperature, or humidity, needlessly complicating all calculations. Lastly, that "percentage humidity," more properly called "degree of humidity," is not really comparable percentage at all. Ten per cent of humidity means that 1 cu. ft. of mixture contains one-tenth the amount of water which a cubic foot of another mixture containing less air would carry if saturated under the same conditions of temperature and pressure.

While such a standard of humidity may be satisfactory for meteorological records, practical design in a field thus obstructed was a dangerous thing to turn over to subordinates, and it seemed advisable to begin by getting rid of stumpage and doing some grading. Finally there was marked disagreement in published tables of the essential constants.

The last question was first taken up. The original sources were examined and an entirely new set of calculations made

*An abstract of a paper presented at the Pittsburg meeting of the American Institute of Chemical Engineers. The complete paper is published in Vol. I. of the Transactions of the American Institute of Chemical Engineers.

from vapor tensions to weight of water per cubic foot, using the following formulæ and constants:

Density of air + .04% $CO_2 = \frac{1.293052}{1 + .00367 \times \text{Temp. C.}}$ in kg per cu. m).

Density of water vapor = .62186 × density air.

Ratio of density at partial pressure to density at 760 mm = 1/760 × partial pressure in mm.

Wt. in kg of 1 cu. m of residual air = 1/760 × part. press. × density of air (in kg per cu. m).

Wt. in kg of 1 cu. m of water vapor at part. pressure = 1/760 × 0.62186 part. pressure × density of air.

Specific heat water vapor = .475.

Specific heat of air = .2373.

TABLE I. HUMIDITY TABLE.

Temp. F.	Vapor Tension in Millimeters	Lbs. Water Vapor per Pound Air	Humid Heat in B.T.U.	Humid Volume in Cu. Ft.	Density in Lbs. per Cu. Ft. @ 760 Millimeters		Specific Volume in Cu. Ft. per Lb. of—	
					Dry Air	Sat'd Mix.	Dry Air	Sat'd Mix.
1	2	3	4	5	6	7	8	9
32	4.569	.003761	.2391	12.462	.080726	.080556	12.388	12.414
33	4.746	.0039176	.2392	12.491	.080559	.080377	12.413	12.442
34	4.949	.0040763	.2392	12.521	.080393	.080198	12.439	12.469
35	5.152	.0042435	.2393	12.549	.080231	.080025	12.464	12.496
36	5.358	.0044151	.2394	12.578	.080069	.079852	12.489	12.523
37	5.572	.0045931	.2395	12.606	.079910	.079680	12.514	12.549
38	5.795	.0047781	.2396	12.636	.079748	.079511	12.539	12.576
39	6.023	.0049678	.2397	12.666	.079585	.079343	12.565	12.604
40	6.264	.0051678	.2398	12.695	.079420	.079181	12.590	12.629
41	6.507	.0053703	.2399	12.724	.079272	.079012	12.615	12.655
42	6.764	.0055849	.2400	12.754	.079113	.078849	12.640	12.682
43	7.026	.0058025	.2400	12.784	.078954	.078677	12.666	12.710
44	7.301	.0060320	.2401	12.813	.078797	.078514	12.692	12.736
45	7.582	.0062670	.2403	12.843	.078641	.078348	12.718	12.763
46	7.874	.0065103	.2404	12.872	.078485	.078178	12.741	12.791
47	8.177	.0067636	.2405	12.904	.078329	.078011	12.766	12.819
48	8.486	.0070239	.2407	12.935	.078172	.077844	12.792	12.846
49	8.811	.0072942	.2408	12.968	.078019	.077678	12.817	12.874
50	9.140	.0075697	.2409	12.999	.077867	.077511	12.842	12.901
51	9.488	.0078620	.2410	13.030	.077714	.077344	12.868	12.927
52	9.841	.0081579	.2411	13.062	.077562	.077178	12.893	12.957
53	10.210	.0084683	.2413	13.095	.077409	.077016	12.919	12.984
54	10.589	.0087865	.2415	13.127	.077258	.076851	12.944	13.012
55	10.980	.0091163	.2416	13.159	.077109	.076685	12.968	13.041
56	11.386	.0094583	.2418	13.192	.076926	.076522	12.993	13.068
57	11.801	.0098064	.2420	13.224	.076810	.076363	13.019	13.096
58	12.235	.010165	.2421	13.258	.076661	.076198	13.044	13.124
59	12.674	.010546	.2423	13.292	.076512	.076032	13.070	13.152
60	13.138	.010939	.2425	13.326	.076363	.075865	13.095	13.180
61	13.608	.011338	.2427	13.360	.076215	.075698	13.121	13.210
62	14.100	.011755	.2429	13.395	.076069	.075529	13.146	13.240
63	14.603	.012182	.2431	13.431	.075924	.075362	13.171	13.270
64	15.122	.012625	.2433	13.467	.075799	.075198	13.196	13.298
65	15.660	.013081	.2435	13.501	.075635	.075039	13.222	13.325
66	16.209	.013547	.2438	13.535	.075493	.074882	13.246	13.354
67	16.783	.014043	.2440	13.572	.075347	.074718	13.272	13.384
68	17.363	.014539	.2442	13.609	.075201	.074552	13.298	13.413
69	17.975	.015065	.2444	13.645	.075062	.074387	13.323	13.441
70	18.595	.015597	.2447	13.683	.074921	.074219	13.348	13.471
71	19.242	.016154	.2450	13.720	.074778	.074054	13.373	13.502
72	19.903	.016724	.2452	13.759	.074636	.073899	13.398	13.532
73	20.585	.017312	.2455	13.797	.074496	.073735	13.424	13.562
74	21.289	.017922	.2458	13.837	.074363	.073570	13.449	13.593
75	22.008	.018545	.2461	13.876	.074218	.073405	13.474	13.624
76	22.759	.019198	.2464	13.916	.074079	.073239	13.499	13.654
77	23.517	.019857	.2467	13.956	.073940	.073072	13.524	13.685
78	24.315	.020554	.2470	13.997	.073801	.072909	13.549	13.715
79	25.124	.021260	.2474	14.039	.073663	.072747	13.575	13.746
80	25.965	.021998	.2478	14.081	.073531	.072585	13.600	13.777
81	26.724	.022752	.2481	14.123	.073395	.072419	13.625	13.809
82	27.711	.023532	.2485	14.167	.073258	.072248	13.650	13.841
83	28.625	.024339	.2488	14.211	.073123	.072080	13.675	13.873
84	29.557	.025164	.2492	14.255	.072987	.071913	13.701	13.906
85	30.529	.026026	.2497	14.301	.072852	.071744	13.726	13.938
86	31.519	.026898	.2501	14.347	.072716	.071574	13.752	13.971
87	32.518	.027818	.2505	14.399	.072584	.071404	13.776	14.004
88	33.582	.028750	.2510	14.442	.072453	.071234	13.801	14.038
89	34.667	.029724	.2514	14.490	.072321	.071064	13.827	14.072
90	35.774	.030718	.2519	14.539	.072189	.070894	13.852	14.106
91	36.913	.031747	.2524	14.588	.072058	.070724	13.787	14.139
92	38.087	.032810	.2529	14.638	.071926	.070556	13.903	14.173
93	39.283	.033896	.2534	14.688	.071794	.070390	13.929	14.206
94	40.528	.035029	.2539	14.740	.071664	.070221	13.954	14.241
95	41.784	.036174	.2545	14.793	.071535	.070051	13.979	14.275
96	43.102	.037388	.2550	14.846	.071407	.069878	14.004	14.310
97	44.434	.038603	.2556	14.900	.071279	.069704	14.030	14.346
98	46.824	.039817	.2562	14.956	.071149	.069530	14.055	14.582
99	47.228	.041206	.2569	15.013	.071021	.069356	14.080	14.419
100	48.679	.042557	.2575	15.071	.070894	.069179	14.106	14.455
101	50.171	.043955	.2582	15.131	.070767	.069002	14.131	14.492
102	51.691	.045382	.2589	15.192	.070641	.068825	14.155	14.530
103	53.271	.046876	.2596	15.253	.070514	.068645	14.181	14.568
104	54.865	.048385	.2603	15.313	.070387	.068464	14.207	14.606
105	56.534	.049973	.2610	15.376	.070264	.068288	14.232	14.643
106	58.219	.051582	.2618	15.440	.070141	.068110	14.257	14.682
107	59.968	.053271	.2626	15.506	.070018	.067926	14.282	14.722
108	61.749	.054996	.2634	15.572	.069894	.067745	14.307	14.761
109	63.578	.056634	.2643	15.640	.069751	.067568	14.333	14.800
110	65.459	.058613	.2651	15.711	.069647	.067383	14.358	14.840
111	67.374	.060492	.2660	15.783	.069525	.067192	14.384	14.883
112	69.359	.062456	.2670	15.854	.069403	.067007	14.409	14.924
113	71.362	.064442	.2679	15.930	.069281	.066823	14.434	14.965
114	73.455	.66540	.2689	16.006	.069167	.066636	14.459	15.008
115	75.569	.068662	.2699	16.084	.069040	.066447	14.484	15.050
116	77.757	.070881	.2710	16.164	.068920	.066253	14.509	15.094
117	79.986	.073150	.2720	16.245	.068800	.066061	14.535	15.137
118	82.273	.075514	.2732	16.328	.068681	.065871	14.560	15.179
119	84.621	.077921	.2743	16.413	.068562	.065675	14.585	15.225
120	87.010	.080402	.2755	16.499	.068443	.065477	14.611	15.272
121	89.479	.082995	.2767	16.592	.068325	.065280	14.636	15.319

Temp. F.	Vapor Tension in Millimeters	Lbs. Water Vapor Per pound Air	Humid Heat in B. T. U.	Humid Volume in Cu. Ft.	Density in Lbs. per Cu. Ft. @ 760 Millimeters		Specific Volume in Cu.Ft. per Lb. of—	
					Dry Air	Sat'd Mix.	Dry Air	Sat'd Mix.
1	2	3	4	5	6	7	8	9
122	91.978	.085622	.2780	16.679	.068207	.065082	14.661	15.365
123	94.581	.088398	.2793	16.777	.068090	.064884	14.686	15.412
124	97.207	.091208	.2806	16.870	.067973	.064689	14.711	15.460
125	99.924	.094147	.2820	16.968	.067857	.064480	14.736	15.509
126	102.688	.09716	.2835	17.069	.067741	.064277	14.762	15.558
127	105.518	.10027	.2849	17.172	.067625	.064075	14.788	15.606
128	108.428	.10350	.2864	17.278	.067510	.063870	14.813	15.657
129	111.382	.10680	.2880	17.386	.067394	.063663	14.838	15.708
130	114.437	.11022	.2896	17.499	.067280	.063449	14.863	15.761
131	117.516	.11368	.2913	17.612	.067166	.063234	14.888	15.814
132	120.724	.11742	.2931	17.730	.067052	.063022	14.914	15.867
133	123.960	.12121	.2949	17.851	.066938	.062809	14.939	15.921
134	127.304	.12514	.2968	17.976	.066826	.062591	14.964	15.977
135	130.702	.12917	.2987	18.103	.066713	.062374	14.989	16.032
136	134.182	.13335	.3006	18.235	.066601	.062158	15.015	16.088
137	137.749	.13768	.3026	18.370	.066489	.061939	15.040	16.145
138	141.372	.14122	.3047	18.508	.066377	.061717	15.065	16.204
139	145.114	.14678	.3070	18.653	.066267	.061487	15.090	16.264
140	148.885	.15150	.3093	18.800	.066156	.061255	15.116	16.325
141	152.807	.15652	.3117	18.952	.066045	.061022	15.141	16.387
142	156.762	.16161	.3141	19.108	.065935	.060790	15.166	16.450
143	160.841	.16696	.3166	19.270	.065823	.060554	15.192	16.512
144	164.986	.17245	.3192	19.437	.065712	.060318	15.217	16.576
145	169.227	.17816	.3219	19.609	.065601	.060082	15.242	16.643
146	173.569	.18409	.3247	19.787	.065498	.060842	15.267	16.710
147	177.976	.19017	.3276	19.971	.065390	.060599	15.293	16.780
148	182.522	.19659	.3307	20.161	.065283	.060353	15.318	16.846
149	187.103	.20310	.3338	20.354	.065176	.059106	15.343	16.919
150	191.860	.21005	.3371	20.559	.065068	.058865	15.368	16.993
151	196.654	.21710	.3404	20.767	.064962	.058605	15.394	17.068
152	201.595	.22455	.3440	20.987	.064856	.058349	15.419	17.141
153	206.610	.23221	.3476	21.211	.064750	.058092	15.444	17.215
154	211.739	.24019	.3514	21.445	.064644	.057833	15.469	17.291
155	216.983	.24854	.3553	21.687	.064539	.057570	15.494	17.370
156	222.305	.25713	.3594	21.936	.064425	.057305	15.520	17.450
157	227.786	.26623	.3637	22.201	.064328	.057036	15.545	17.533
158	233.308	.27546	.3681	22.468	.064224	.056767	15.570	17.616
159	239.034	.28542	.3728	22.754	.064120	.056493	15.596	17.701
160	244.803	.29553	.3776	23.045	.064016	.056218	15.621	17.788
161	250.742	.30627	.3827	23.382	.063913	.055938	15.646	17.877
162	256.767	.31737	.3880	23.670	.063810	.055656	15.671	17.968
163	262.923	.32900	.3936	24.001	.063707	.055372	15.697	18.060
164	269.213	.34123	.3994	24.349	.063605	.055084	17.722	18.154
165	275.592	.35385	.4054	24.708	.063502	.054795	15.748	18.250
166	282.155	.36733	.4118	25.090	.063400	.054500	15.772	18.349
167	288.764	.38106	.4183	25.478	.063298	.054203	15.798	18.449
168	295.609	.39602	.4254	25.900	.063198	.053901	15.823	18.545
169	302.504	.41125	.4327	26.300	.063098	.053598	15.848	18.654
170	309.593	.42756	.4405	26.790	.062997	.053292	15.874	18.761
171	316.781	.44472	.4485	27.267	.062899	.052983	15.899	18.875
172	324.120	.46257	.4570	27.768	.062799	.052670	15.924	18.986
173	331.612	.48162	.4660	28.301	.062699	.052353	15.949	19.101
174	339.207	.50140	.4754	28.855	.062599	.052032	15.975	19.218
175	347.015	.52285	.4856	29.454	.062500	.051708	16.000	19.339
176	354.873	.54472	.4960	30.063	.063402	.051382	16.025	19.462
177	363.003	.56901	.5076	30.738	.062304	.051049	16.050	19.589
178	371.190	.59386	.5194	31.428	.062207	.050715	16.076	19.719
179	379.598	.62097	.5322	32.180	.062112	.050378	16.101	19.851
180	388.121	.64942	.5458	32.967	.062015	.050035	16.126	19.987
181	396.815	.67985	.5602	33.810	.061915	.049693	16.152	20.126
182	405.686	.71265	.5758	34.717	.061816	.049338	16.177	20.269
183	414.674	.74703	.5921	35.666	.061719	.048985	16.202	20.415
184	423.904	.78519	.6102	36.718	.061623	.048626	16.227	20.566
185	433.194	.82430	.6288	37.796	.061529	.048265	16.253	20.719
186	442.793	.86911	.6501	39.029	.061434	.047898	16.278	20.879
187	452.456	.91535	.6721	40.302	.061339	.047529	16.303	21.041
188	462.374	.96731	.6967	41.729	.061244	.047153	16.328	21.209
189	472.422	1.0227	.7230	43.251	.061149	.046774	16.353	21.381
190	482.668	1.0835	.7519	44.918	.061053	.046391	16.379	21.557
191	493.111	1.1510	.7843	46.768	.060961	.046003	16.404	21.739
192	503.690	1.2230	.8190	48.739	.060867	.045611	14.429	21.925
193	514.545	1.3067	.8579	51.021	.060774	.045213	16.454	22.119
194	525.368	1.3933	.8990	53.402	.060681	.044813	16.480	22.315
195	536.744	1.4994	.9494	56.302	.060588	.044405	16.505	22.521
196	548.093	1.6108	1.0023	59.349	.060496	.043997	16.530	22.730
197	559.731	1.7436	1.0714	62.978	.060403	.043586	16.556	22.945
198	571.519	1.8912	1.1358	67.024	.060311	.043169	16.581	23.166
199	583.531	2.0629	1.2171	71.696	.060219	.042742	16.606	23.393
200	595.771	2.2680	1.3147	77.304	.060127	.042308	16.631	23.638
201	608.155	2.4968	1.4231	83.537	.060036	.041868	16.656	23.886
202	620.871	2.7987	1.5665	91.768	.059946	.041425	16.682	24.142
203	633.657	3.1189	1.7186	100.499	.059855	.040982	16.707	24.401
204	646.845	3.6509	1.9159	115.210	.059764	.040530	16.732	24.675
205	660.116	4.2272	2.1562	131.028	.059674	.040075	16.758	24.954
206	673.718	4.9776	2.5681	151.272	.059585	.039610	16.783	25.248
207	687.489	6.0698	3.1238	180.906	.059495	.039143	16.808	25.549
208	701.516	7.6748	3.8948	224.623	.059406	.038673	16.834	25.860
209	715.805	11.1295	6.2569	362.091	.059317	.038049	16.858	26.289
210	730.267	15.8174	15.9148	562.054	.059228	.037323	16.884	26.796

Kg per cu. meter × .062428 = lb. per cu. ft.

Results of these calculations are given for every degree Fahrenheit in Table I (Humidity Table).

These results (Col. 3, Table I) differed by as much as 4 per cent from the various published figures, but showed a surprising agreement (¼ per cent due to allowance or 0.04 per cent CO_2) with the figures of the United States Weather Bureau. The reason for this was inquired into. Quoting from the Psychrometric Tables of the United States Weather Bureau of 1900 (prepared by Dr. C. F. Marvin, Professor of Meteorology), it will be seen how carefully the Weather Bureau's tables were prepared:

"The vapor pressures employed were derived from Broch's reduction of Regnault's observations. All the values given by these tables at temperatures below the freezing point are noticeably higher than Regnault's observations. In view of this systematic discordance and the further circumstance that Regnault's experiments did not include observations at the extremely low temperatures frequently recorded at Weather Bureau stations, the writer (Dr. Marvin), in 1891, made a new determination of the maximum pressure of aqueous vapor at low temperatures.

"In the course of these experiments it was found an easy matter to reduce the temperature of the water employed many degrees below 32° Fahr., without freezing it, and in these cases the vapor pressure was higher than the pressure from ice at the same temperature. Independent experiments in Sweden, by Julius Johlin, at about this time, led to the same results.

"A comparison of the vapor pressures derived from the several sources mentioned shows that Broch's computed values at low temperatures do not agree at all well with Regnault's experiments, from which they are derived, whereas the experimental results of Regnault, Johlin, and the writer (Dr. Marvin), agree very closely.

"At temperatures below 32° Fahr., therefore, it has been considered necessary to reject Broch's values, and the vapor pressures over ice, as deduced from the writer's experiments, have been used in the calculation of the tables. At temperatures above 32° Fahr. the values taken from Broch's tables are employed. At 32° Fahr. the value for the vapor pressure found by the writer from the mean of a large number of experiments was identical with that of Broch; hence, there is no break in the continuity of the two tables at the point of junction."

The calculated weight of water per pound of air does not agree with actual determinations, but shows progressive differences, possibly owing to association of molecules in the vapor, which are too small in magnitude, however, to affect our present purpose.

Regarding this difference Dr. Marvin writes:

"In all ordinary circumstances it is assumed that the expansion and contraction of partially saturated aqueous vapor is in accordance with the same laws as apply to air and ordinary gases, which do not easily condense to the liquid state.

"The adopted density of saturated aqueous vapor is not determined directly from experiment, but is deduced theoretically from the observed fact that two volumes of hydrogen and one of oxygen combine to produce two volumes of water vapor.

"The weights of unit volumes of hydrogen, oxygen, and dry air are accurately known, from which the specific gravity of aqueous vapor is found to be 0.6221."

The compared figures from various sources were submitted to Dr. Marvin, but he replied that, "The Weather Bureau has not attempted to measure the quantity desired directly; in fact, the difficulty of experimentally maintaining a condition of absolute saturation and avoiding supersaturation, or mechanically suspended moisture on the one hand and superheating on the other, is so great as to perhaps render the results thus obtained less acceptable than the theoretical values."

We must, therefore, accept these figures as generally recognized throughout the United States and at least as well established as our purpose requires.

New Definitions of Humidity, etc.

In the next place a graphic method of calculation was desired, for rapidity and freedom from gross error, much more dangerous than slight inaccuracy. Also it was sought to visualize as far as possible what actually occurred to simplify comparison. A number of trials proved it far more satisfactory to disregard entirely the question of volume until it finally became necessary to calculate cross-sections of air ducts, passages, etc.

A great gain was made by regarding "humidity" as "the pounds of water carried by 1 lb. of air (or equivalent weight of other gas having the same volume) when saturated at a given temperature and pressure," and regarding the "percentage humidity" as "the weight of water actually carried by a pound of air divided by the maximum which could be carried by that pound of air under the same conditions."

This suggested as a substitute for specific heat something which we called the "humid heat," namely, "the b.t.u. required to raise 1° Fahr. 1 lb. of air, plus whatever water vapor it might carry when saturated at the temperature and pressure," and a "percentage humid heat" varying regularly with the "percentage humidity" between that of air alone and that of the saturated mixture corresponding to the same conditions.

Similarly we obtained a new "humid volume," i. e., the volume of 1 lb. of air when saturated under known conditions, and a corresponding "percentage humid volume" varying with the percentage of saturation between the specific volume of 1 lb. of dry air and the volume of 1 lb. of air saturated under the same conditions. We use the usual "density" plotted as shown and a "percentage density" (which does not vary proportionally with percentage humidity) only for calculating velocities in $V = k\sqrt{h\frac{d-d}{D}}$, for gravity circulation plants where k is always uncertain and damper regulation can be provided.

It has been simply delightful to find the ease and facility with which calculations could be carried out when dealing with conditions thus defined.

Given a mixture (let us say saturated 90°), all we need to do for the heat capacity is to pass horizontally across to the "humid heat" curve (see chart I accompanying this issue as a supplement) and if we want the heat capacity of any unsaturated mixture, we find the corresponding temperature and percentage of saturation, dropping vertically downward (merely a change in temperature) until we reach the saturation curve and then horizontally across to the heat capacity of this saturated mixture. If we want the volume or density of any saturated mixture it is directly given by horizontally crossing to the "humid volume" or "density" curves.

One difficulty (not yet overcome in any other way so far as the writer has been able to learn) is presented by the question: "How much moisture will a saturated (or unsaturated) mixture, which has been heated to a certain temperature and is used for the drying of any material, take up before it becomes saturated (or 90 per cent or 80 per cent saturated, if we prefer) and to what temperature will it fall?" It is necessary, at present, to assume the specific heat of water vapor as constant,[1] 0.475, though there is both theoretical and experimental evidence that under these conditions it acts as superheated steam with a variable specific heat.

But assuming it constant, the law of cooling is obviously

$$\frac{dx}{dt} = \frac{1.438\,S}{1601 - t}$$

in which x represents the weight of water per pound of air, t represents the temperature (Fahrenheit) at which evaporation is stopped, and S the specific heat of the mixture which we are heating and which is to be cooled by the evaporation it causes. But we have not any similar equation to express the line of saturation, 90 per cent saturation or any other degree, and we cannot solve our problem mathematically. The latest German

[1] "Specific Heat and Total Heat of Superheated Steam," Geo. A. Orrok, "Power," Aug. '04, p. 426.

works on this subject give us nothing but a clumsy method of interpolation which requires a preliminary calculation of cumbersome tables and repeated trial calculations to select values which will approximately satisfy the two requirements.

We can, however, represent the direction of this "cooling by evaporation" graphically by the "lines of cooling." It is apparent that no matter what the maximum temperature may be, if the condition of our preliminary mixture is represented by a point on any one of the curves of percentage saturation, the action of heating it is fully portrayed by elevating that point directly upward to the line of maximum temperature (neither its specific heat nor "humid heat" is changed).

The direction of its fall in cooling dx/dt will be the same no matter whether saturated at 75°, or 50 per cent at 96°, dependent upon its original air and water components expressed by the distance from the axis of Y (as governed by the numerator $1.148\,S$) and upon the temperature where it finally ends up in the course of its cooling (as governed by the denominator $1601\text{-}t$). In considering the cycle it is perfectly correct to assume all the evaporation to have taken place at this final temperature.

We have, therefore, drawn for various final temperatures on our diagram a series of straight diagonal lines on the same Humidity Curve Sheet representing dx/dt for various initial mixtures, the water content of which is given by the distance from the Y axis where the left-hand end of each line ceases. On the left-hand side of the curve, owing to the slight difference of inclination of corresponding lines for successive final temperatures, it is necessary to draw only one of these fan-shaped sets of lines and these need have been drawn only for each successive 10° of temperature.

Here the saturation curves show that the temperature varies rapidly and water content (with corresponding specific heat) varies slowly. On the right-hand portion of the curve, where the variation of "water absorbed" (and of specific heat) is greater, they are needed every 5°.

The lines for intermediate points can be drawn if desired, but it will be noted that a difference of 5° produces a difference of value of dx/dt of about ½ of 1 per cent.

Now, it is apparent that with these lines upon our curve we may take any given mixture which is to be heated (whether saturated or not to start with), run up the vertical to the maximum allowable temperature and selecting from the nearest group of lines of cooling that one which starts from the nearest position with reference to the Y axis, we may follow down parallel to this to the saturation curve.

If we are not considering the complete saturation, but only the 70 per cent saturation of the final exit mixture (before cooling and condensation) this is only a first approximation. Upon noting at what temperature the diagonal intersects the curve of 70 per cent saturation we must then redraw our incline, parallel to the incline corresponding to this mixture for this nearest final to the one thus discovered.

Example Showing Use of Curves.

As an illustration of the use of these curves, assume 50 lb. per minute of water to be removed at a temperature not exceeding 110° Fahr. with available air at 80° Fahr and 90 per cent saturated. Assume the material to be so distributed and exposed that the air will leave the dry room at 80 per cent saturation. How much air will be required? What will be its final temperature, etc?

On the chart we find 80° Fahr., ordinate crosses the 80 per cent humidity dotted curve at an abscissa of 0.0200 lb. water per pound of air. If heated to 110° the change of condition is represented by increased ordinate with no change of abscissa (absolute water content), the point rising vertically to 110°, where we find it between 30 per cent and 40 per cent humidity curves, or approximately 34 per cent saturated.

On exposure to the material being dried, the air will cool and take up moisture, changing its condition parallel to the inclined straight line to the immediate right of the point, 0.0200 lb. 110°.

This diagonal shows that it will be 50 per cent saturated at 101°, 70 per cent saturated at 96°, and 80 per cent saturated at 90°, with an abscissa of 0.0248 lb. water per pound of air, having picked up $0.0248 - 0.0200 = 0.0048$ lb. water per pound of air.

To remove 50 lb. water per minute would require $50 \div 0.0048$, or about 10,000 lb. of air per minute. To ascertain the heat required for this entering air we require the heat capacity of air at 80°, 90 per cent saturated.

Starting from the same point as before, we drop to the 100 per cent saturation curve, and find that our mixture has cooled to 77¼° Fahr., and passing on the horizontal of 77¼° toward the left, to the Humid Heat curve, we find that each pound of air requires 0.248 b.t.u. per degree Fahrenheit to heat it and its water of saturation. We have then $0.248 \times (110 - 85°) \times 10{,}000 = 62{,}000$ b.t.u. per minute.

To find the volume of 10,000 lb. of air, let us say at the exit, where each pound is carrying 0.0248 lb. of water with it and is 80 per cent saturated, we have only to pass across from the point 80 per cent, 0.0248 lb., 90° Fahr. (where we ended the first operation), along the horizontal of 90° to the Specific Volume and the Humid Volume curves. The specific volume of dry air at 90° is 13.86, that of saturated air (Humid Volume) 14.55, and 80 per cent of this difference is $0.69 \times 0.8 = 0.552$ added to the volume of dry air $= 13.86 + 0.55$, or 14.41 cu. ft. per pound $= 144{,}410$ cu. ft. air per minute at the exit.

The lines at present shown upon the sheet have amply sufficed for all purposes of practical design and have made it possible almost at a glance to select the conditions best suited to a given problem. They show vividly, for instance, the great advantage secured by applying heat between successive partial saturations by progressive exposure to the material being dried —also the enormous economy of restricted ranges of temperature.

Finally, we are developing therefrom, with little labor or calculation, a series of curves showing for every 10° Fahr. entrance temperature, the maximum conditions of efficiency, assuming various saturations of air before and after passing over the material, and a locus of maximum efficiency for these entrance and exit assumptions.

Heat Losses from Pipes.

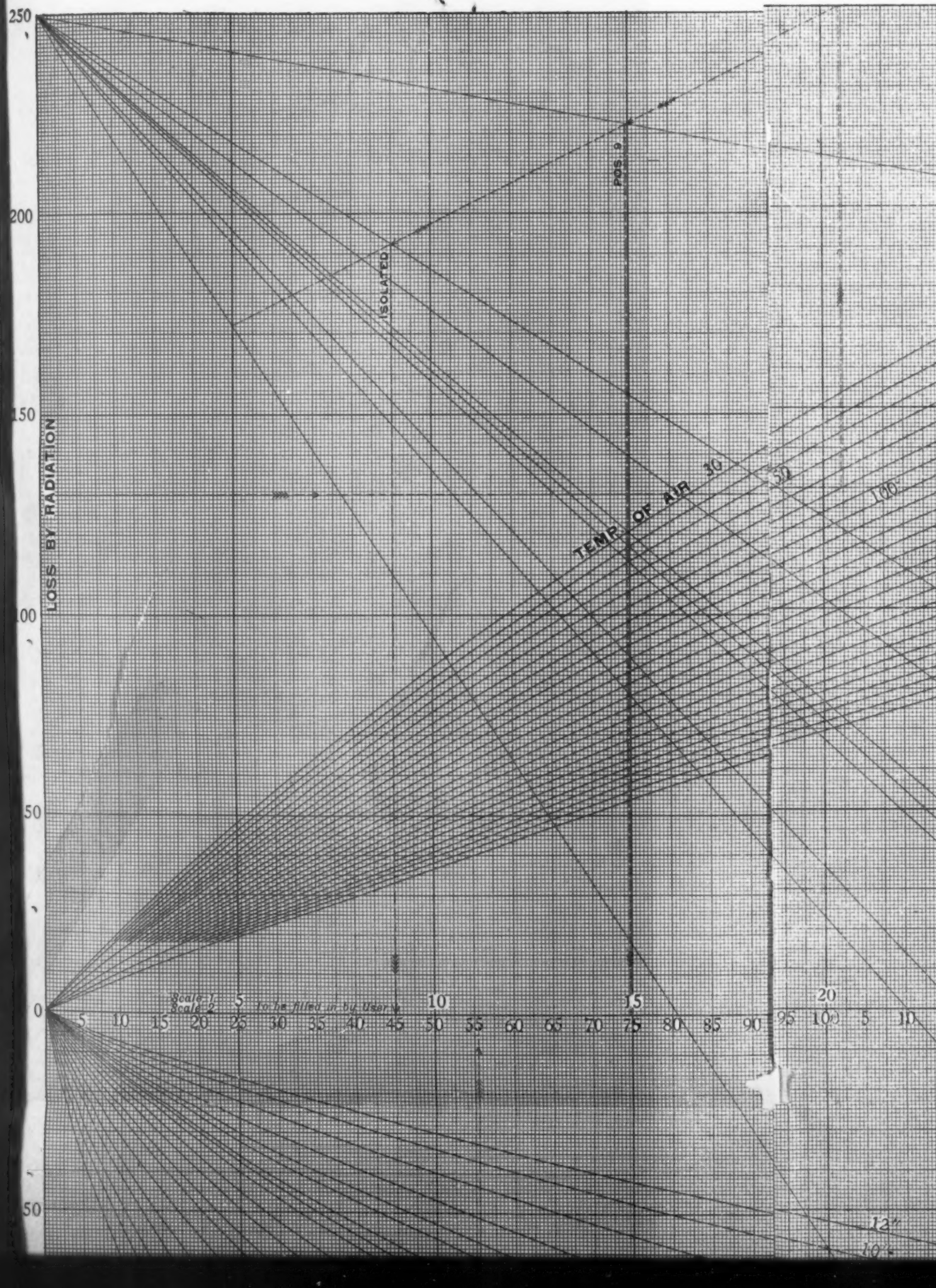
The auxiliary curves (see chart II: Loss of Heat from Pipes, accompanying this issue as a supplement) are intended to simplify calculations of heating pipes and present a graphic solution of the law of Dulong and Petit, as modified by Peclet and Box. The use of the curves is extremely simple and is shown by arrows. They enable us to obtain the two components, radiation and convection loss, from pipes ⅛ in. to 12 in. U. S. standard size, per foot of length singly or in any of the 10 positions which they may occupy in a bank.

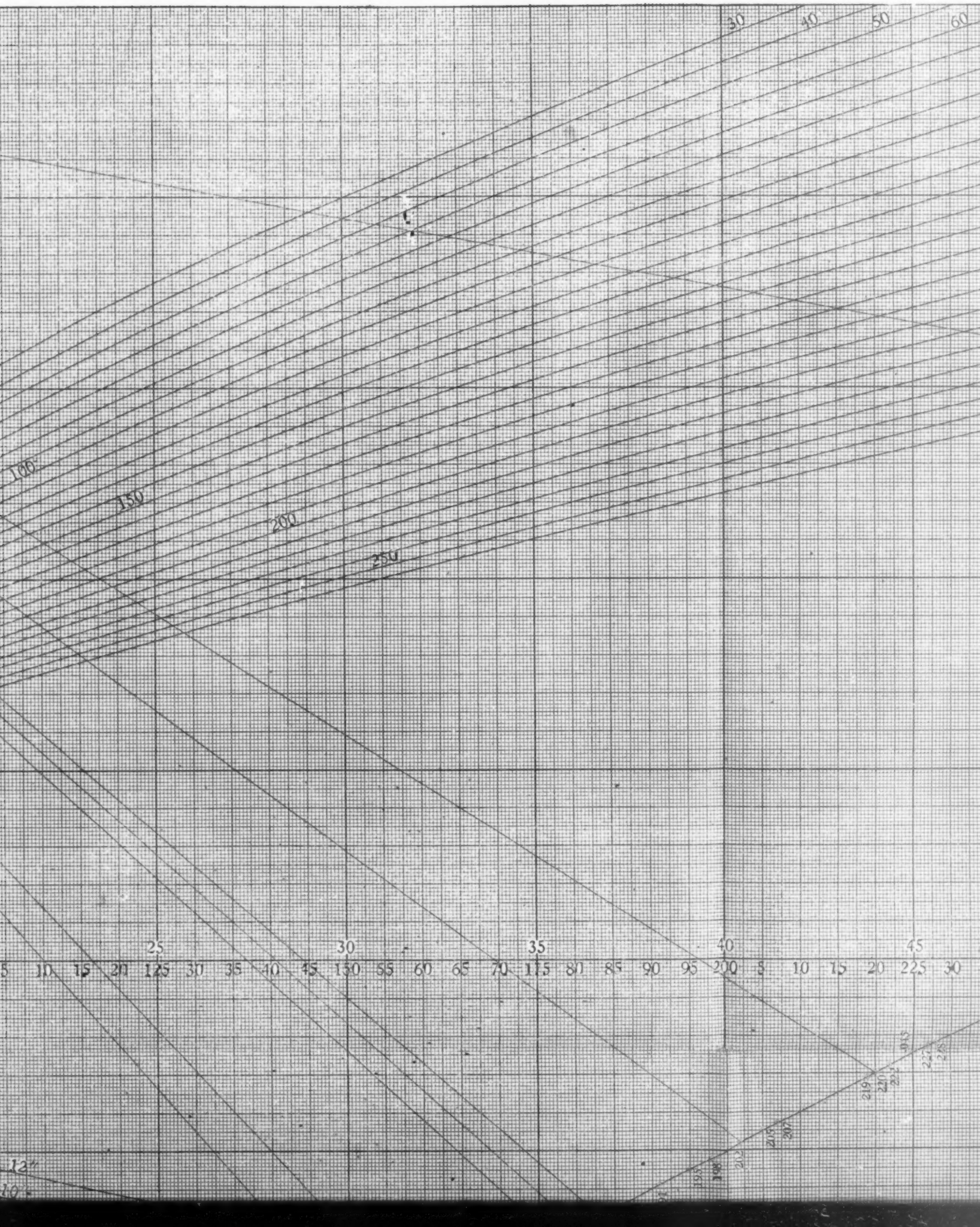
The fundamental basis of these calculations (0.64 b.t.u. radiation and 0.95 b.t.u. convection) are not accurate by several per cent, and cannot remain constant, as the pipes rust or gather dust. No effort, therefore, has been made to secure greater accuracy than given in the figures of Box. A safety factor has been added to the result given by the curves and the size of this factor is being gradually reduced by experience.

In our designs hitherto, the velocities of gases across the heating pipes has always been practically that due to natural circulation, and no provision is shown here for variable velocities. It can easily be added by a single inclined line if desired.

(The original paper of Dr. Grosvenor contains here several tables of figures representing the derivation of the curves of Chart III. These tables are here omitted, since the results are given completely in Chart II.)

The losses by convection for each standard size pipes are plotted with temperature difference as ordinates and b.t.u. per hour as abscissæ in the lower half of the curves of Chart II.


250
200
150
100
50
0
50
LOSS BY RADIATION
ISOLATED
POS. 9
TEMP. OF AIR 30
32
100
Scale-1
Scale-2
to be filled in by User
5 10 15 20 25 30 35 40 45 50 55 60 65 70 75 80 85 90 95 100 5 10
5 10 15 20
12"
10"


30
40
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150
200
250
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30
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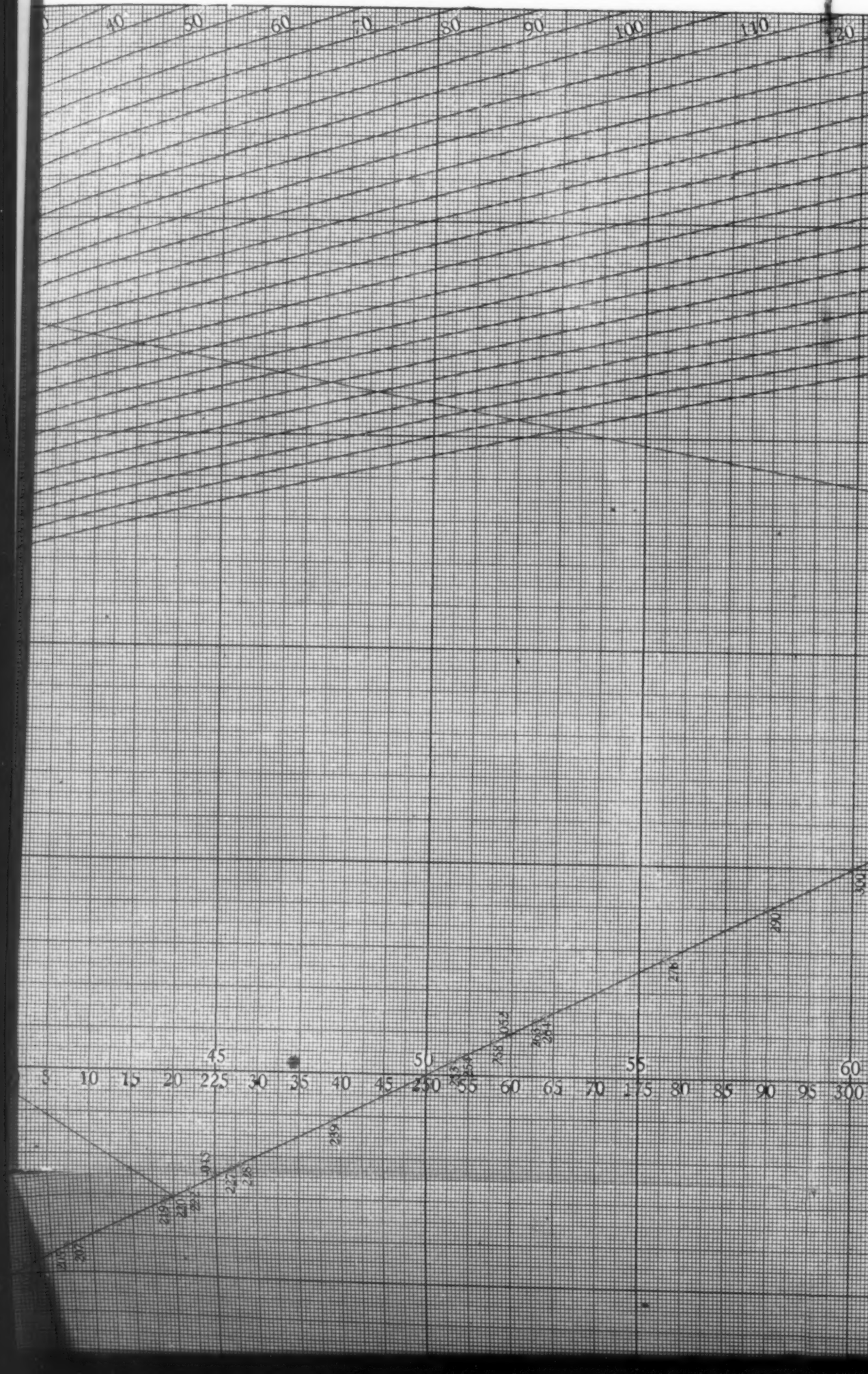
40
45
5 10 15 20 25 30 35 40 45 50 55 60 65 70 75 80 85 90 95 00 5 10 15 20 25 30
12"


40
50
60
70
80
90
100
110
120
45
50
55
60
3
10
15
20
225
30
35
40
45
250
55
60
65
70
175
80
85
90
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300
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262
245
226
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230
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284
268
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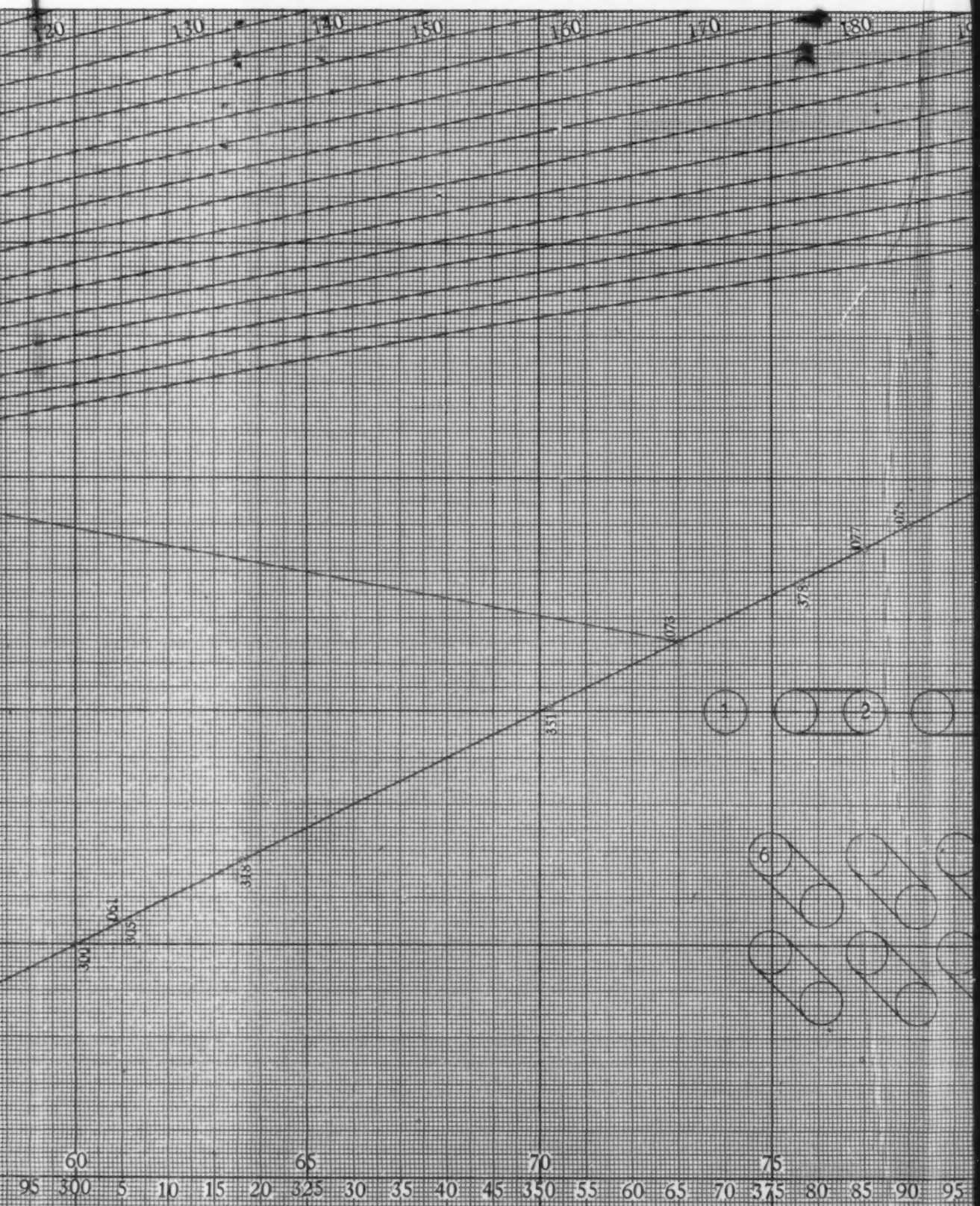
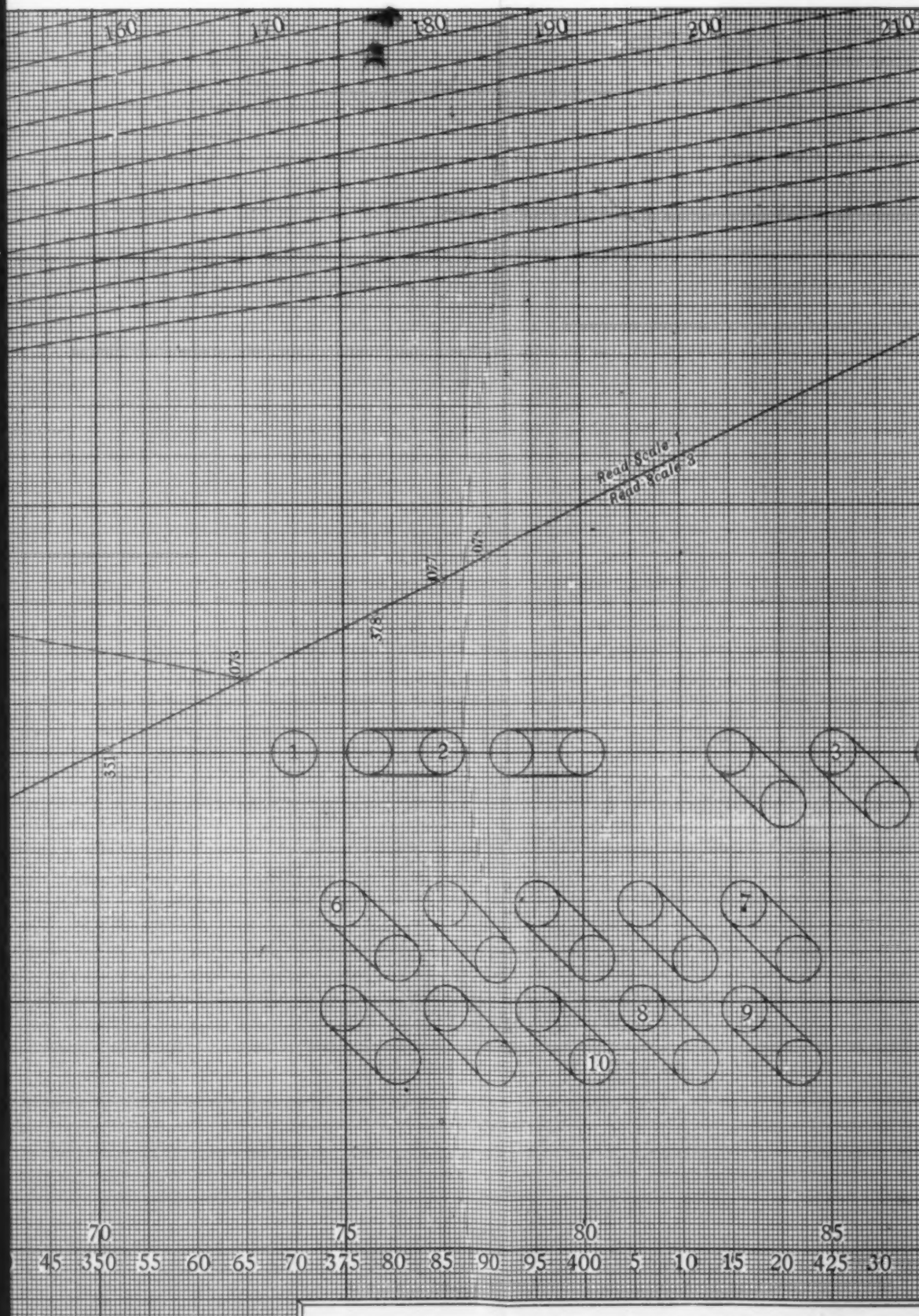
256
254
260



ISOLATED FACTOR = (

POSITION FACTOR = (

U. S. Pipe	Isolated	Pos 1	2	3
1/8	.068			
1/4	.090			
3/8	.113			

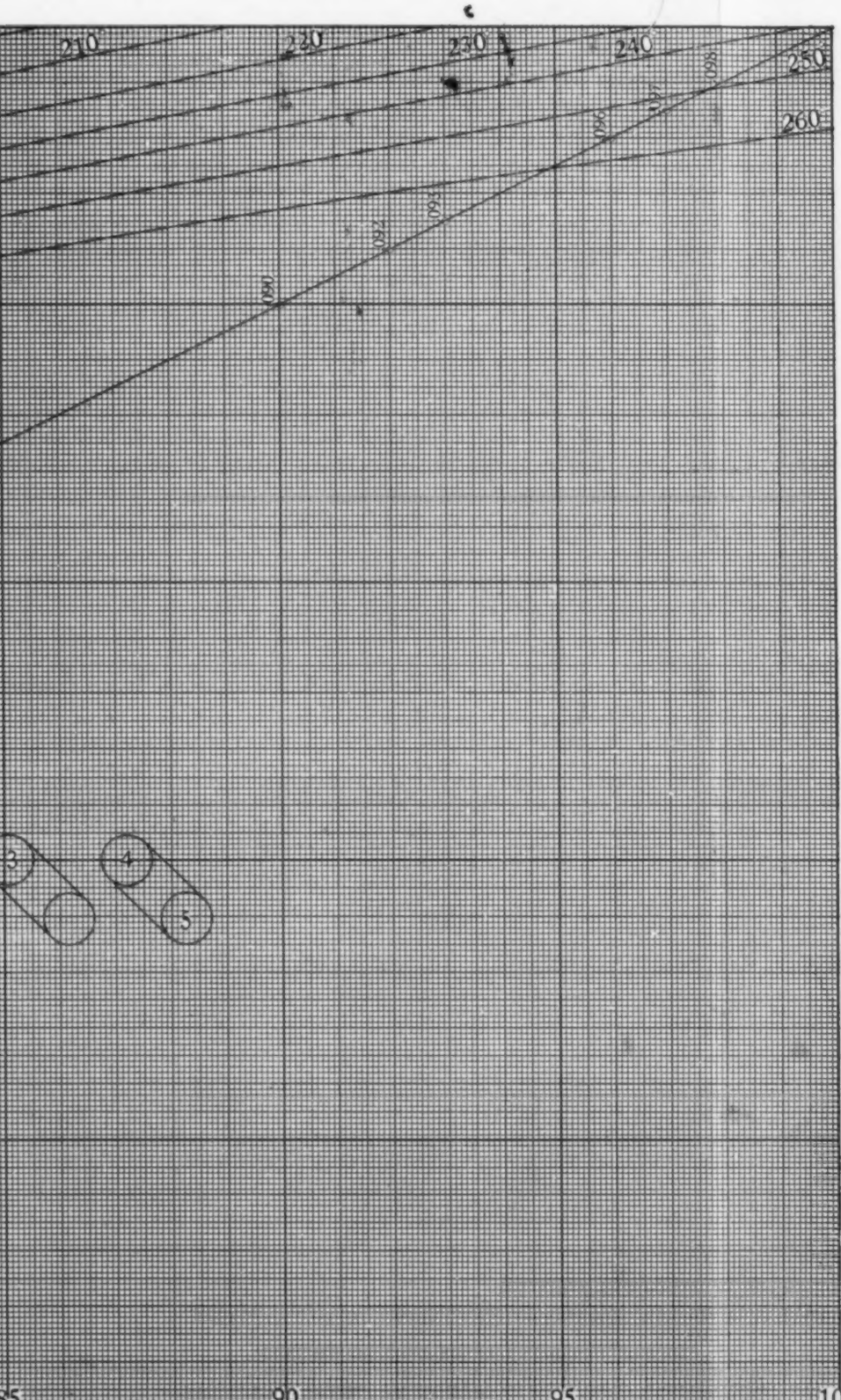


Pipe Size and Location Factors

ISOLATED FACTOR = 0.64 B.T.U. X external surface of pipe 1 ft. long

POSITION FACTOR = 0.64 B.T.U. X ext. surf. of pipe 1 ft. long X ang

U. S. Pipe	Isolated	Position of pipe in assembled coils (numbered as in									
		1	2	3	4	5	6	7	8	9	10
⅛	.068										
¼	.090										
⅜	.113										
½	.140	.127	.115	.090	.098	.112	.096	.077	.022	.045	.061



85 90 95 100 Scale 1
25 30 35 40 45 450 55 60 65 70 475 80 85 90 95 500 Scale 3
Scale 2

B.T.U.
PER FT.
PER HR.

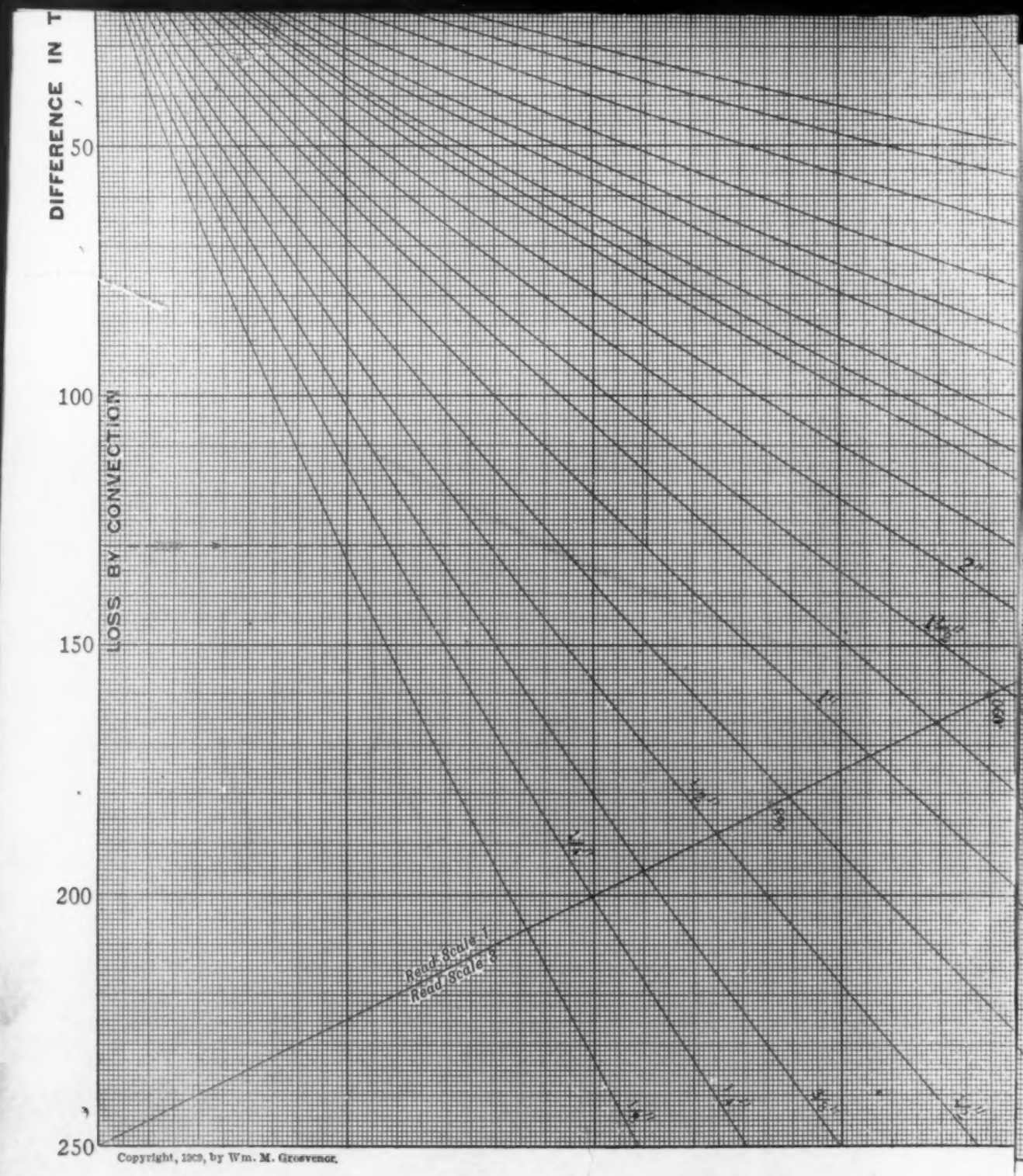
ctors

ipe 1 ft. long

long X angle of free radiation ÷ 360

mbered as in diagram above)

			10				
				U. S. Pipe	Isolated		
.061				4	.754		
				4½	.838		



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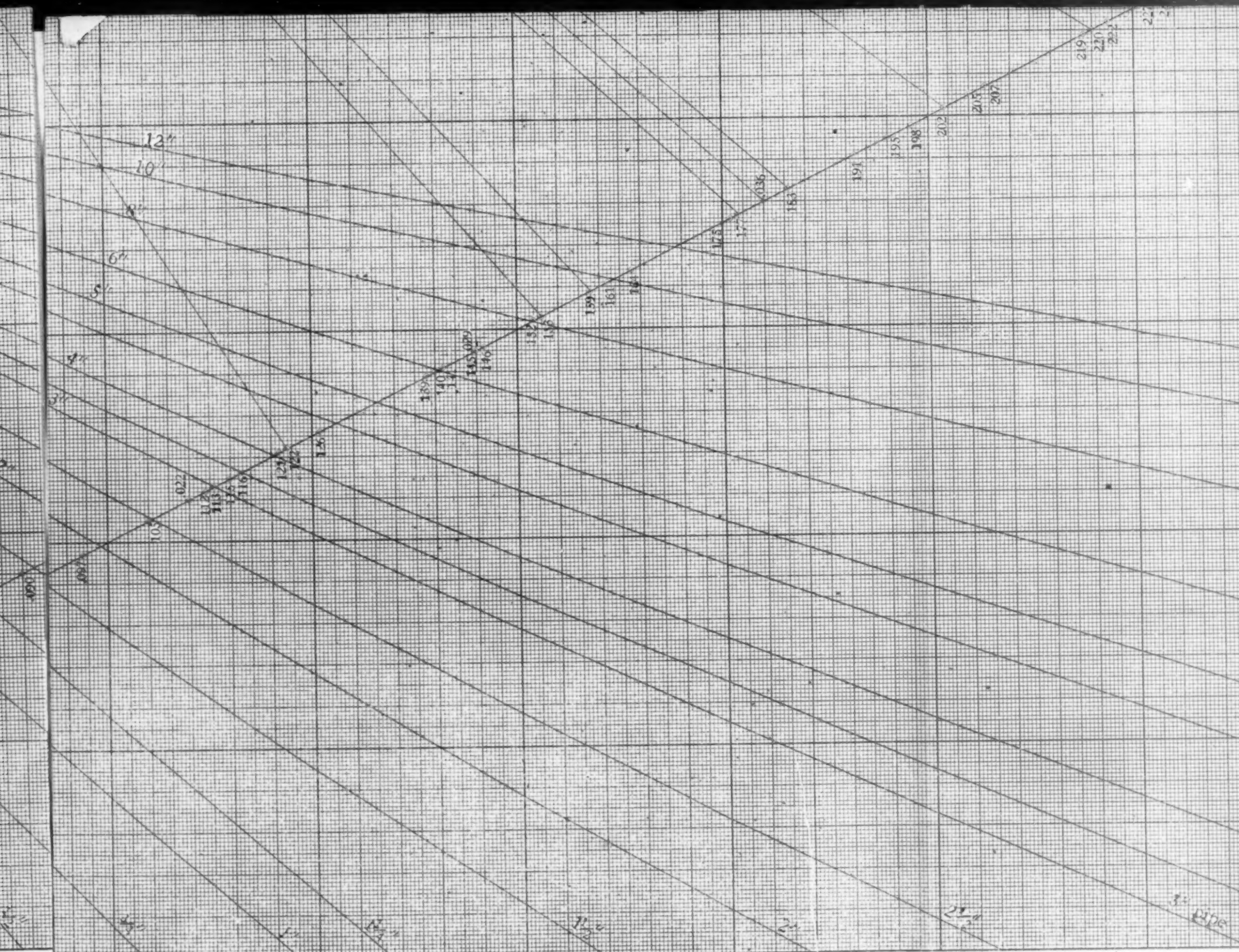


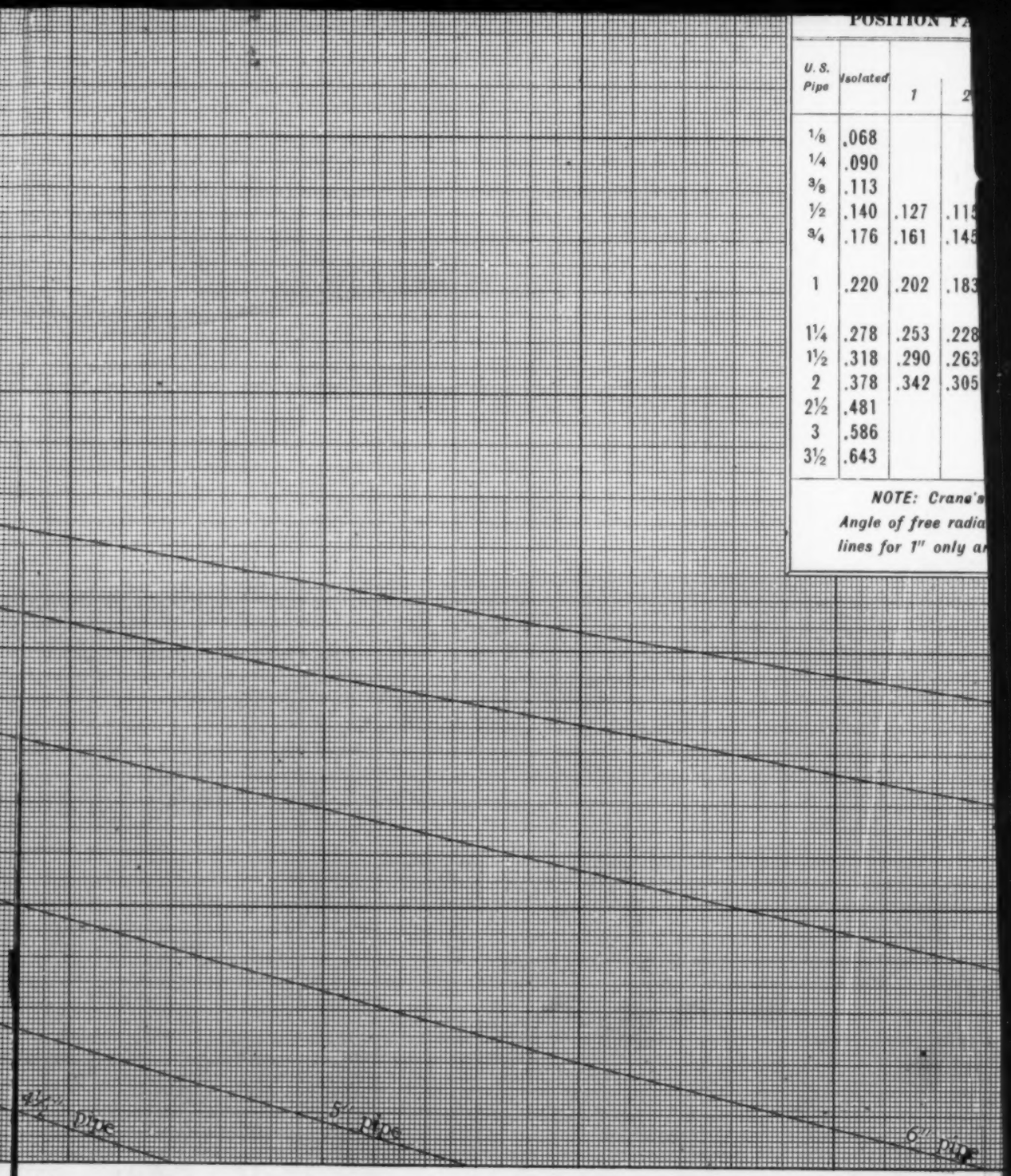
CHART II: LO

RT II: LOSS OF HEAT FROM PI

POSITION FA

U. S. Pipe	Isolated	1	2
1/8	.068		
1/4	.090		
3/8	.113		
1/2	.140	.127	.11
3/4	.176	.161	.14
1	.220	.202	.183
1 1/4	.278	.253	.228
1 1/2	.318	.290	.263
2	.378	.342	.305
2 1/2	.481		
3	.586		
3 1/2	.643		

NOTE: Crane's
Angle of free radia
lines for 1" only a

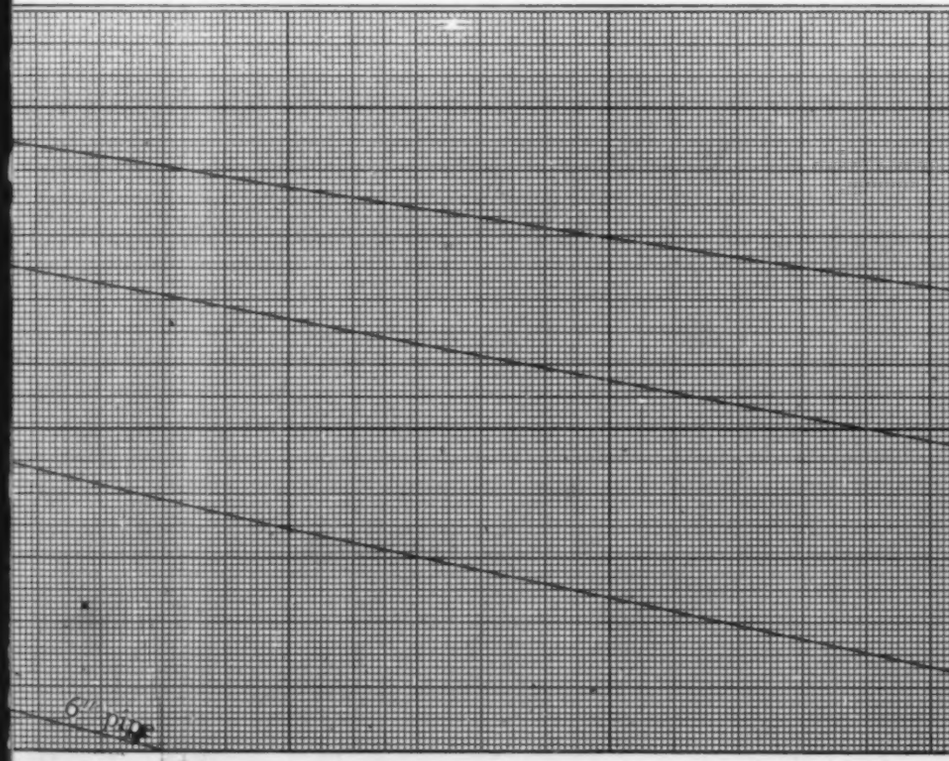


PIPES

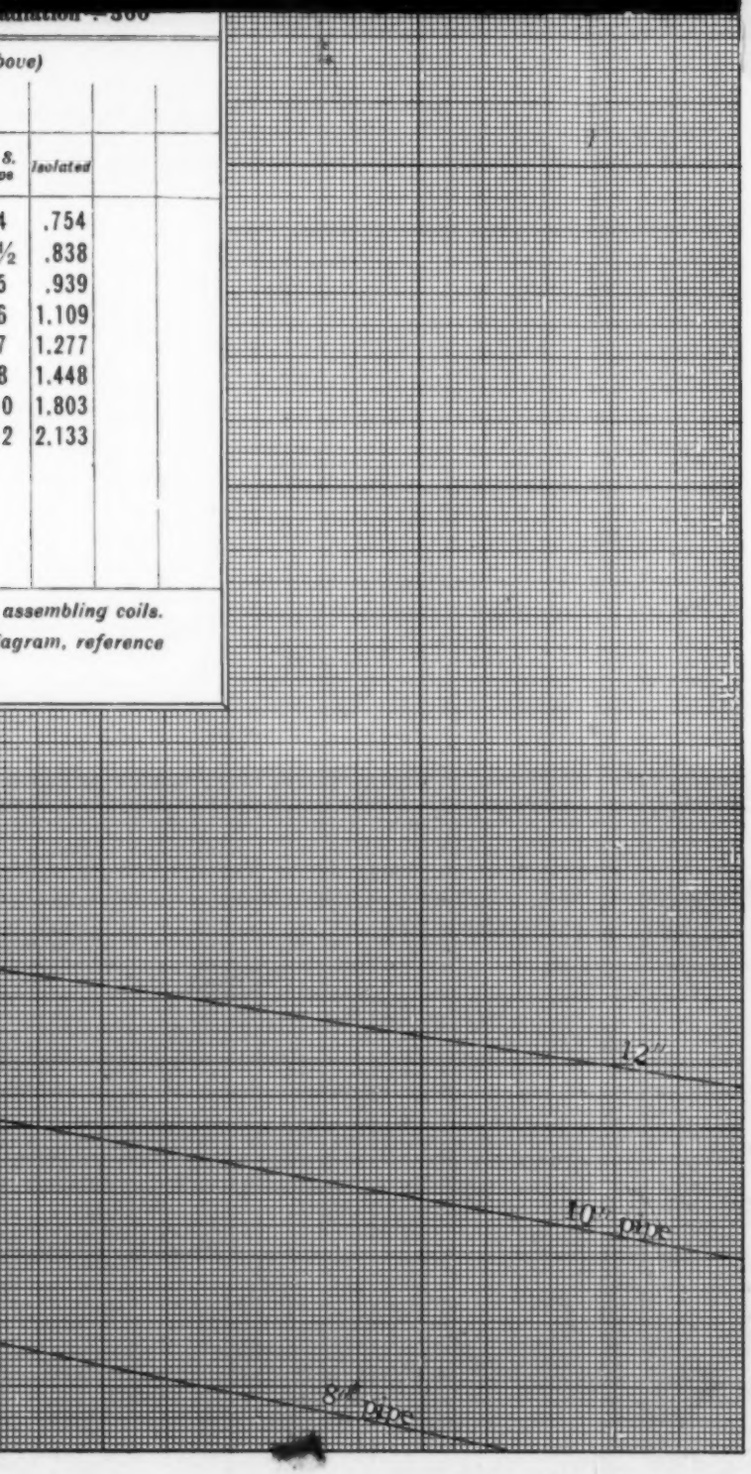
SITION FACTOR = 0.64 B.T.U. × ext. surf. of pipe 1 ft. long × angle of free radiation ÷ a

ted	Position of pipe in assembled coils (numbered as in diagram above)													
	1	2	3	4	5	6	7	8	9	10				
													U. S. Pipe	Isolated
													4	.754
0	.127	.115	.090	.098	.112	.096	.077	.022	.045	.061			4½	.838
6	.161	.145	.116	.126	.141	.122	.097	.029	.058	.078			5	.939
													6	1.109
0	.202	.183	.146	.159	.177	.153	.121	.036	.073	.097			7	1.277
													8	1.448
8	.253	.228	.180	.198	.222	.191	.152	.045	.090	.122			10	1.803
8	.290	.263	.207	.227	.254	.219	.175	.052	.103	.139			12	2.133
8	.342	.305	.239	.264	.300	.258	.205	.058	.116	.164				
1														
6														
3														

NOTE: Crane's standard malleable return bends wide pattern are assumed used for assembling c
le of free radiation would differ for other pattern. To avoid complicating the diagram, refer
for 1" only are drawn.



Accompanying "Calculations



	Isolated		
4	.754		
½	.838		
5	.939		
6	1.109		
7	1.277		
8	1.448		
0	1.803		
2	2.133		

assembling coils.
agram, reference

Calculations for Dryer Design" by WILLIAM M. GROSVENOR.

The presentation of radiation is somewhat complicated by the fact that the correction factors vary both with temperature and with difference of temperature. These correction factors are represented by the fan-shaped set of curves on the upper half of the diagram.

It is frequently desired to use pipes in banks wherein position greatly affects radiation, though not influencing convection. Modified factors reduced in proportion to the radiating angle obscured by the other pipes of the bank are, therefore, calculated (assuming Crane's Standard malleable return bends are used for assembling) for the various possible positions indicated by Nos. 1 to 10 on the diagram accompanying the chart. These various factors for the isolated and for the various partly obscured positions are given on the chart. The point to which the diagonal for this graphic multiplication should be drawn is indicated on the main guide line, but only a few of the diagonals to these points have been drawn, as our own particular interest centers mainly on combinations of 1-in. pipe. The other lines can easily be drawn when required.

The simplified calculation of wall, ceiling and window losses was found to be well worked out in curves drawn by Mr. Burt S. Harrison and published in the *Heating and Ventilating Magazine* of November, 1907.

Similarly the capacity and drop in pressure in steam mains has been carefully worked out by E. C. Sickles and published in *Power*.

The design of the condenser remains for consideration. It is a surprising fact, but a most thorough search has revealed no data applicable to this case. We use banks of horizontal 1-in. pipe with water or brine circulating within them. If they were merely cooling air we could find ample data for design. They would transmit about 1.5 b.t.u. per square foot per degree per hour. If they were condensing practically pure steam, some quite recent work would be available. They would transmit from 150 to 300 b.t.u. per square foot per hour. But they do neither and yet they do both. A formula has been developed upon purely theoretical grounds similar to those of the law of mass action. It will be submitted later (not as a solution, but as a view for criticism and test), together with examples showing how far it is justified by actual practice.

Notes on Electrochemistry and Metallurgy in Great Britain.

(From our Special Correspondent.)

Societies and Institutions.

The Faraday Society has delayed its opening meeting of the season until November 30th, and the Physical Society's annual exhibition of apparatus has also been fixed for a later date than usual, the 14th of December. The inaugural address of Dr. Gisbert Kapp, president of the Institution of Electrical Engineers, dealt with generators, transformers, power transmission and electric railways, but did not touch on any metallurgical subject. His remarks as to the necessity of educating the potential users of electrical plants are worthy of consideration; but many will share the view expressed in a recent article in the *Times* that "it is to be regretted that Dr. Kapp, instead of covering such an extensive field in his survey, did not devote his address almost exclusively to the enormous benefits to be obtained through the more rapid application of electricity to textile and other factory purposes."

The Reduction of Some Refractory Metals from Their Chlorides.

In a paper recently presented to the Chemical Society by Messrs. J. Norman Pring and W. Fielding, the authors have described their investigations of the reactions by which certain refractory metals may be prepared by the action of electrically heated carbon rods or filaments on their volatile chlorides. When tungsten chloride was used without admixture, under diminished pressure, decomposition occurred at temperatures between 1200° and 1500°; and with molybdenum chloride, below 1300°, adherent deposits of the pure metals being formed on the carbon. At higher temperatures carbides were produced; but when small amounts of aqueous vapor were present the metals were reduced free from carbide. Pure crystalline silicon was obtained at temperatures between 1700° and 1850° by addition of hydrogen to the vaporous chloride. At first a thin layer of carbide was formed; and this prevented carbonization of the subsequent deposit until the temperature was raised to 1900°, when carbide alone was formed. In the case of boron, the carbide B_6C, was always deposited at temperatures between 1800° and 2000°. Approximate photometric determinations of the radiating power of the metal-coated carbon showed a very great increase in luminosity for a fixed consumption of power, amounting with tungsten and molybdenum to nearly three times the radiation from carbon rods. Deposits of silicon and of boron carbide caused only a very slight increase.

A New Use for Electrolytic Hypochlorite.

Dr. F. W. Alexander, medical officer of health for Poplar, has been carrying out a series of experiments on the purification of the water in the borough swimming baths by the addition of electrolytic disinfecting fluid. The large first-class swimming bath has a capacity of 75,000 gallons, and it was found that the addition of 100 gallons of electrolytic hypochlorite solution in four installments—40 gallons on the Saturday, 40 gallons on the following Monday, 10 on Tuesday night and 10 on the Wednesday—had the effect of rendering the water perfectly clear and fit for use, whereas the bath is usually emptied and cleaned on Tuesday and fresh water run in. After a total of 899 persons had used the bath on Wednesday the water became slightly clouded, but the bottom of the bath was distinctly visible. On the Thursday night after 1159 persons had used the bath the water was discolored and the bottom somewhat obscured. The superintendent reported that the effect of the addition of the electrolytic fluid was to preserve the water in as good a condition after five days' use as it would have kept in one day's use only without such addition; and the secretary of a swimming club stated that the members felt fresher, not languid, and that they had lost the usual "dead feeling" on coming out of the bath.

One of the laundresses employed in cleaning the swimming costumes reported that those articles had no unpleasant odor since the fluid had been used, but previously they gave off a most unpleasant smell. The chemical and bacteriological report by a Fellow of the Institute of Chemistry on the bath water without hypochlorite shows that the chlorine present as chlorides was 2.5 grains per gallon as against 2 grains per gallon present in the East London water supplied. Some nitrites were present; the amount of oxygen required to oxidize the organic matter present was 0.1 grain per gallon, which is almost the figure given by an ordinary slightly polluted water. Qualitative tests showed the presence of B. Coli communis, Streptococci, and Staphylococci; but B. Enteritidis Sporogenes was not present. Quantitative tests showed approximately 10,000 organisms per cc. capable of growing at blood heat, and 2000 capable of growing at ordinary temperatures. The preponderance of the former indicates that they are chiefly of animal (human) origin. Samples of the water to which hypochlorite had been added were found to contain no nitrites; free chlorine estimated at 0.5 mg. per litre; and after incubation for four days no organisms were found growing in the cultures made, which result might have been anticipated from the fact that free chlorine was present.

In the course of these experiments a peculiar fact came to light, namely, that the water used by women and girls was always dirtier than that used by the same number of men and boys; and after investigation the conclusion was arrived at that this was due to dust accumulated in the women's long hair. The other baths of the Corporation were also treated with hypochlorite, and the same satisfactory results were obtained; and

the masters and mistresses who attended with the boys and girls from the schools said the water was fresher, more invigorating, and had lost its body odor. It is proposed to use about 38 gallons of electrolytic fluid to the bath of 75,000 gallons. The cost of this quantity would be about 55 cents.

For some time past the public health officials of the borough of Poplar have been using electrolytic hypochlorite instead of carbolic acid for general disinfecting purposes, and have found it combines higher efficiency with considerably reduced cost.

Company Meetings.

The Castner-Kellner Alkali Company.

This company's fourteenth ordinary general meeting was held on November 25th, under the presidency of the Chairman, the Rt. Hon. G. W. Balfour, who congratulated the shareholders on the financial statement. The net profit—but without deduction for debenture interest or depreciation and reserve—was £134,154, which was an increase of more than £18,000 on last year's profit. This result was more satisfactory than appeared at first sight, because this year was by no means a prosperous one in the chemical trade, whilst the increase arose in almost every product which the Company turned out, which must be taken as indicating the broad basis of prosperity on which the company rested. Deducting debenture interest and the interim dividend paid in May, the net available balance is £117,711. On last year's accounts, as in previous years, loss was written off for realization of disused plant. In 1907 the amount was about £16,000. Usually it was a comparatively small sum, but this year it did not appear at all. Losses in respect of realized plant differ from usual depreciation, and this is partly the reason why the sum for depreciation has been raised from £30,000 to £35,000. On suspense account the sum of £1,370 had been written off, and as a consequence that account disappeared. A suspense account had been necessary because investigations were inevitable which it would not be fair to charge to revenue of a single year; but now that the general reserve fund was established it was to be hoped that the suspense account would not be revived. The general reserve fund was started last year, and it is proposed to carry £25,000 thereto, making its total amount £37,000 which is, in fact, a cash reserve, invested in good, easily realizable securities, and is available for the purpose of a suspense account or for dividends in less prosperous years. This arrangement being agreed to, there was a sufficient balance to pay a dividend of 9 per cent, making with the May interim dividend 14 per cent for the year, and leaving £15,840 to carry forward. The chemical trade seemed to be improving, and there was no reason to anticipate a worse result for the current year's accounts than the last year's had shown.

The report and accounts were received and adopted, and a vote of thanks to the Chairman closed the proceedings.

The Aluminium Corporation, Ltd.

A scheme for the transfer of the assets of this company to a new company was sanctioned in the Companies' Court on November 23d by Mr. Justice Swinfen Eady. The petition stated that the company was incorporated with a capital of £500,000, of which £440,000 were in 7 per cent participating cumulative preference shares and £60,000 were in ordinary shares with a non-cumulative preference of 7 per cent, whilst 40 per cent of the remaining profits went to the preference shares and 60 per cent to the ordinary. The £1 preference shares issued were 257,205, of which 252,222 were fully paid. There was a debenture issue of £26,906, and a large unsecured debt. The company was wound up, and liquidators were appointed in 1908; and in May of that year a scheme of arrangement was sanctioned, but was not carried out through lack of money; nor was an attempt to realize the company's assets successful. The proposed scheme provides for the payment of the general creditors, and unsecured creditors for less than £10, in full; unsecured creditors for more than £10 would receive preference shares in the new company, and shareholders would get part-paid shares. There was no opposition; and the scheme was sanctioned, subject to slight amendment and the consent of shareholders to accept shares in the new company.

London, Dec., 1909.

Synopsis of Periodical Literature

Iron and Steel.

Electric Steel Furnace of Girod.—The chief interest in an illustrated article on the Girod furnace, by Professor W. Borchers, in *Stahl und Eisen*, of Nov. 10, lies perhaps in the somewhat personal remarks in which he expresses his enthusiasm concerning this type, which he considers to be "the simplest of the present commercial electrode steel furnaces." The keynote of the invention of Girod he finds in the way in which the steel bath is made a resistor, producing itself Joulean heat. Since the Girod furnace has been described and illustrated quite often in this journal (for instance, in our Vol. VII, pp. 9, 259, 323, 371, 458) and since Professor Borchers' article contains little that is new, only a few data on the new Girod electric steel works in Ugine (first described in our Vol. VII, p. 9, and of great interest, as being the first large and entirely electric steel works) may be given here, as Borchers has recently visited them. At present there are installed in these works three furnaces, each of two tons capacity, and two larger ones, each of 10 to 12 tons capacity. Castings up to 20,000 kg weight can be produced. Other electric steel furnaces of Girod are in use, or in course of erection, at the works of Oehler & Company, Aarau, Switzerland; John Cockerill Company, in Seraing, Belgium; Stotz & Company, Stuttgart, Germany; Stahlwerk Becker, Krefeld, Germany; Gutehoffnungshütte, Oberhausen, Germany; Ternitzer Stahlwerk von Schoeller, Ternitz, Austria; Danner & Company, Judenberg, Austria; Hungarian State Steel Works, Diosgyor, Hungary. (A Girod furnace is also reported to be in course of erection in a steel plant in this country.) The same article by Borchers was printed in *Metallurgie*, Nov. 8. The issue of Dec. 8 of this journal contains a letter by the Elektrostahl G. m.b.H., Lindenberg, which criticises various points in Borchers' article and endeavors to show the superiority of the Héroult furnace over the Girod furnace. The same issue contains a reply by Professor Borchers in which he states his reasons for considering the Girod furnace to be the superior one. Both letters are of a rather personal nature.

Armature Steel and Permanent-Magnet Steel.—A suggestive study of steel for the electrical industry is given by G. Mars in *Stahl und Eisen*, Oct. 27 and Nov. 10. For armatures and transformers a steel is wanted which has a high permeability and a low hysteresis loss—that is, metallurgically speaking, a steel which has a high content of ferrite and a coarse structure indicating a loose agglomeration of the molecules (so as to permit them to follow rapidly the reversals of magnetic polarity). This condition is fulfilled by a steel containing 2 to 3.5 per cent silicon and a very small amount (less than 0.1 per cent) carbon. The requirements of steel for permanent magnets are, of course, quite different. Here high remanence is chiefly desired. In carbon steels the remanence increases with increasing carbon content up to 0.97 per cent C. and then decreases. The hardness increases and reached a maximum at 0.8 to 0.9 per cent C. and then remains practically constant. The maximum of remanence has been found by Mme. Curie at 1.20 per cent, but the author's figure of 0.97 per cent C. seems more probable. Tungsten steels are, however, far superior to carbon steels for permanent magnets. The best steel for permanent magnets at present

used contains about 0.60 per cent C. and 5 per cent tungsten. This superiority of tungsten steels over carbon steels is due to the presence of more free iron in the former with equal hardness. A tungsten steel of the stated composition gets highest remanence if the hardening temperature is 930° C. to 950° C. The paper contains interesting suggestions on the relation between metallurgical structure and magnetic properties.

Effect of Titanium on Steel.—An interesting paper on this subject is published by ED. VON MALTITZ in *Stahl und Eisen* of Oct. 13. In his introductory remarks he mentions some experiments made five years ago in a basic open-hearth furnace of the Dominion Iron & Steel Company in Sydney, Canada, where he used titaniferous iron ore (containing 40 per cent Fe and 16.8 per cent titanic acid) as an addition for the purpose of increasing the fluidity of the basic slag and improving the desulphurization. Metallic titanium did not enter thereby into the steel bath, but the fluidity of the slag was considerably increased without detriment to the furnace lining; a distinct influence on the removal of sulphur was also observed. The experiments were, however, not carried far enough to reach final conclusions. The author discusses in his article especially the use of ferro-titanium for steel refining. Since this matter has been discussed at length in an article by Mr. C. V. Slocum, in our Vol. VII, p. 128, and in another article on the results obtained with titanium-treated steel rails (Kessler's curve, etc.) in our Vol. VII, p. 498, we mention only the new points in Mr. Maltitz's article. Ferro-titanium, containing from 10 to 15 per cent Ti is preferably used. It is added after the steel has been recarburized and after ferro-maganese or ferro-silicon has been added. Care is to be taken that the ferro-titanium is not mixed with the slag. It should be dissolved in the steel bath itself for the purpose of removing oxygen and especially nitrogen, the resulting compounds passing into the slag. Reference is made to the detrimental effect of nitrogen on steel according to the work of H. Braune (our Vol. V, p. 51). Maltitz found in converter steel for rails 0.013 to 0.014 per cent nitrogen. When the same steel was treated with titanium, the nitrogen content was reduced to 0.004 or 0.005 per cent. This is a very important result, since "nitrogen should be considered to be at least as detrimental as phosphorus." Some figures from practice are given showing that a very homogeneous steel is obtained by means of treatment with titanium. In most cases an addition of ½ per cent of ferro-titanium containing 10 to 15 per cent Ti is sufficient. While this improvement of the quality of steel is important, it should be remembered that a material which cools quietly and without developing gases, tends to the formation of pipes, like crucible steel, if care is not taken to cast at a moderate heat. This precaution is, therefore, important. Steel treated with titanium cools much more slowly than ordinary steel; the reason for this is not clearly understood. But it is also a fact that articles of steel which have been treated with titanium are heated less than ordinary steel under otherwise equal conditions; for instance, cast-iron or steel rolls, which have been treated with a small amount of titanium, can be machined more quickly than ordinary steel rolls, on account of the smaller increase of temperature due to machining. As was noticed in our former articles, some remarkable results have been obtained with titanium-steel rails on difficult sections of railroads. These are reviewed at some length by Mr. Maltitz.

Oxy-Acetylene Welding of Sheet Iron.—In *Stahl und Eisen* of Nov. 17, H. RINNE reports on the results obtained in practice by the sheet-rolling mill of the Schulz Knaudt Company in Essen, who have used for many years the water-gas welding process. On account of its reliability and cheapness they continue to use this method for longitudinal cylinder seams. During the last years they have also employed oxy-acetylene welding, which has been found especially useful for repairs of leaks developed in seams produced by the water-gas process. The main advantage of oxy-acetylene welding in comparison with water-gas welding is that the former process permits to make the seam at will thicker and stornger than the iron sheet itself. Some care must be taken, however, in oxy-acetylene welding; the acetylene must be pure and must not contain dust or impurities; the dimensions of the burners must correspond to the thickness of the sheets; the additional steel to be put on to the weld must not be of inferior quality and should not be added until the edges to be welded together are thoroughly fused. Of main importance are the hammering of the weld and the annealing after the weld is completed. Finally, wherever it is possible, the two sheets should be welded together from both sides. If carefully carried out, the results obtained with oxy-acetylene welding are at least as good as with water-gas welding. That seams made by oxy-acetylene welding are very reliable was proven by hydraulic-pressure tests made with hollow receptacles. The published statement that only thin sheets (up to 8 mm with oxy-hydrogen and up to 10 mm with oxy-acetylene) can thus be welded is wrong, since the author has used oxy-acetylene welding with best success with sheets of 30 mm or more. There is no danger of oxidizing the material during the welding operation, if care is taken to use a reducing flame (an excess of acetylene or hydrogen), but this is quite easy.

Precious Metals.

Precipitation by Zinc.—In an interesting paper read before the Chemical, Metallurgical and Mining Society of South Africa (*Journal* of the Society, March, 1909), Mather Smith claims that the generally accepted principle is wrong that it is necessary to have 1 cu. ft. of zinc shavings for the precipitation of one ton of solution in 24 hours; he further claims that more zinc than is necessary is being used, that an excess of zinc is both harmful and costly, that precipitation boxes are often constructed on a wrong principle, and that a comparison of the rate of the flow of solution per cubic foot of zinc, which does not also include or consider the difference in area and number of compartments in the precipitation boxes, is misleading. He gives examples from his practice to prove that the superfluous zinc does a great deal of harm, "as it must occasion an unnecessary consumption of cyanide and charge the solutions with an excess of zinc, which in circulating through the plant will in a great measure be deposited as white precipitate at the heads of the boxes." The principle on which he recommends to construct precipitation boxes is to allow one square foot in area for the precipitation of one ton of solution in 24 hours, with a maximum height of 24 inches from the top of the zinc frames to the top of the solution, and with a maximum of five compartments in each box. In the discussion of the paper (April issue of the *Journal* of the Society), J. H. Johnson says the problem resolves itself more into a matter of time the solution is in contact with the zinc than the depth or amount of Zn required to a ton of solution. In the old-style boxes and in the boxes recommended in Smith's paper the time for which the solutions are in contact with the zinc was from 20 to 30 minutes or more, while in many cases now the solution is only in contact for a period of 6 to 8 minutes. Depth and amount of zinc shavings per ton of solution have to be increased when the time of contact decreases.

Zinc Dust Versus Zinc Shavings.—An interesting comparison of precipitation of gold from cyanide solutions by zinc shavings or by zinc dust was given by A. J. Clark in a paper before the Chemical, Metallurgical and Mining Society of South Africa (*Journal* of the Society, Jan., 1909). The comparative results were obtained in the No. 1 sand plant of the Homestake Mining Company on a working scale with solutions practically identical in composition, during 12 months of 1906 with zinc shavings and during 9 months of 1908 with zinc dust. The average strength of solution precipitated by zinc shavings was 0.03 per cent; if below this figure the solution was strengthened by addition of barren solution of 0.07 per cent strength. The average strength of solution precipitated by zinc dust was 0.0242 per cent; the minimum strength encountered and success-

fully precipitated by dust was 0.015 per cent. The principal comparative figures are as follows:

	Shavings.	Dust.	Shavings by dust.
Tons precipitated	5909	5585	
Dollars recovered, total	$1513	$1513	
Per cent. precipitated	91.7	94.4	
All costs, labor and supply	$243.28	$76.93	$166.35
Gold discharged in barren solutions	$126.43	$99.41	$27.04
Value of cyanide in wasted barren solution	$709.00	$540.60	$168.40
Totals	$1,078.73	$716.94	$361.79

Telluride Gold Ore.—The metallurgy of the telluride gold ores of the Cripple Creek district in Colorado has been the subject of a report (edited by Thos. B. Crowe) of the Portland Section of the Chemical, Metallurgical & Mining Society of South Africa. It was published in the May issue of the *Journal* of the society. Besides a large tonnage of ore of exceptional richness, the Cripple Creek district yields ore which, on account of its refractory nature, is left standing in the mine stope or thrown upon the waste dump. The refractory nature of these ores is due largely to the occurrence of the gold in combination with tellurium, forming a compound (AuAg) Te_2. After a trial of other methods, which turned out failures, roasting followed by barrel chlorination and concentration has become the popular treatment. This method works very well as far as the extraction is concerned, and will continue to be used so long as the present grade of ore is maintained; but as the amount of low-grade ore standing in the stopes and upon the dumps becomes larger year after year, it becomes more apparent that a cheaper method of treatment must be found, the problem resolving itself into one of raw treatment at the mine, rather than transporting 50 miles by rail to the cheap fuel in the valley. With this end in view the Portland Gold Mining Company has carried on extensive experimentation.

An interesting feature of these experiments was the use of potassium persulphate, the object being to dissolve the tellurium. When potassium persulphate is alone in solution, it acts as a strong oxidizing agent. But when mixed with cyanide solutions, its oxidizing influence is greatly retarded. It then acts as a "slow oxidizer or depolarizer, thereby greatly increasing the dissolving power of cyanide solutions and when used in small proportions, 1 lb. to 10 lb. per ton of solution, it does not destroy the cyanide to any great extent." Potassium persulphate, when mixed with cyanide solution, completely dissolves the telluride of gold. Three processes, based on this principle, were tried.

(1) A preliminary treatment by an alkaline persulphate, followed by a treatment with a cyanide solution. The results were poor, since extreme amounts of persulphate must be used to obtain a good tailing.

2. A treatment by a solution containing both alkaline persulphate and potassium cyanide. The process was not found to be economical.

3. A treatment by a solution of an alkaline persulphate, an alkaline iodide, and potassium cyanide, contained in one solution. The object was to obtain the reaction $KCy + 2I = ICy + KI$ and thus produce cyanogen iodide (which is an exceedingly good solvent for the telluride of gold) and not to depend directly upon any of the ingredients other than cyanide and cyanogen iodide as solvents. In using this process, good tailings were invariably obtained, even with very weak solutions. But it was found that the solutions after leaving the ore and standing in contact with the air lost their cyanide, especially when heavy sulphide ores were treated. However, on passing through the zinc boxes, the deterioration of the cyanide is stopped, due undoubtedly to the reducing action encountered there. Alkaline persulphate may be produced economically by electrolysis of salt cake. "Potassium iodide, although an expensive chemical, would remain as such in the solution, the persulphate causing it to give up its iodide, this iodide combining with the cyanide to form cyanogen iodide, with the alkali to form an alkaline iodide, or alkaline iodate, but always finally reverting back to an alkaline iodide, as any iodate formed would be reduced to iodide in the zinc box."

The report is discussed in the July issue of the *Journal* by D. J. Moir and C. H. Cropper and in the September issue by T. B. Stevens. Moir criticises some of the chemical statements in the report, while C. H. Cropper gives some notes on successful practice with telluride ores at Kalgoorlie, W. A. There is a wide difference in the solubility of the gold in telluride minerals. The ore being treated at Kalgoorlie is for the most part a sulphotelluride. In the early days of the field, when the grade of ore was high and the extractions low, it was discovered that tellurium was a factor to be reckoned with in future treatment. Bromo-cyanide enabled better extractions to be obtained, although the residues contained high values. At the present day the average grade of ore has fallen considerably; the highest may be taken at 55s, the lowest at 28s. In the constant effort to reduce the value of the residues the economic factor has prevailed, inasmuch as an extraction of 85 per cent on a refractory ore may show a better cost sheet than a 95 per cent. As to which of the two processes in vogue is the better, viz., dry crushing or wet crushing, is a matter of opinion, but the facts are that with the dry crushing and all roasting plants the residues are under 1 dwt. without the use of BrCy, while the wet crushing residues are 1½ dwt., the raw slime only being treated with BrCy. Total treatment costs, as they are to-day, range from 8s. 8d. per ton to 10s. 8d. for wet crushing, from 10s. up to 11s. 8d. per ton for dry crushing. The capital outlay on a dry crushing plant is greater than for wet crushing, but the difference would be compensated for in the lower residues obtained; greater improvements in roasting furnace design have advanced dry crushing practice. With the wet crushing plant, concentration of the mineral is employed, but there is a proportion of very fine mineral which is carried away in the slime and which is the cause of the higher values in residues.

T. B. Stevens asks whether the authors have made any comparative experiments, using bromo-cyanide instead of potassium cyanide and persulphate, and if so, whether the latter method gives a very much better extraction. The cost of the latter would be higher. The cost of chemicals for bromo-cyanide treatment at Kalgoorlie has always been one great disadvantage of an otherwise successful process. The writer's experiments with Cripple Creek ores showed that a good extraction could be obtained by bromo-cyanide treatment, fine grinding being a sine quâ non. At Kalgoorlie it is found most economical to take out the pyritic portion of the ore by concentration, and treat it separately by roasting and cyaniding; otherwise, in treating this very refractory material together with the bulk of the ore, the cost for bromine salts, to get a good extraction, is found to be excessive. With Cripple Creek ores the writer found this also to be the case, and, in the latter district, he believes that the small amount of high-grade concentrates could be more profitably shipped to the smelters instead of treating it at the mine.

Continuous Collection of Sand for Cyaniding.—A paper on an interesting new development in gold metallurgy on the Rand was presented by W. A. Caldecott before the Chemical, Metallurgical and Mining Society of South Africa and is printed in the August issue, 1909, of the *Journal* of this society. The new process is the outcome of long-extended experiments, the object of which was to effect some saving in the very high capital expenditure for a modern cyanide plant. On consideration it appeared that the most likely prospect of success lay in eliminating the sand collecting vats, or rather in utilizing them for treatment purposes instead of merely as sand storages, which take the place of the old tailing dams. The upper vats of the superimposed type of sand plant were on their first introduction before the Boer war utilized as treatment vats as well as sand collectors, but the results of contamination of mill water with cyanide caused this practice to fall into disuse, and

in any case a considerable time elapsed between the filling of a vat and its charge yielding gold-bearing solution to the boxes. Hence, although it reduced cost of transfer to a minimum, this type of plant fell into disuse as being expensive, through having only one-half its capacity available for extraction purposes, and was replaced by separate blocks of collectors and leachers (with belt or truck transfer) in the ratio of 1 to 2 or 2 to 5, which is now the common practice on the Rand. On considering how to utilize sand-collecting vats for treatment purposes and to retain their advantages, including minimum ground space required, while avoiding the difficulties mentioned, the problem became essentially that of continuously removing the water and slime from the tailing pulp so as to leave the same in proper condition for cyanide treament. The solution was found in the use of conical classifiers in connection with a specially designed slowly rotating horizontal vacuum filter. The first filter of this kind, constructed by the author, was 10 ft. in external diameter and consisted of an annular launder 12 in. wide containing a filter cloth upon a grating as a false bottom. The filtering area was thus 28.3 sq. ft. The space under the filter cloth was connected by means of radial pipes to a central hollow spindle, in which a vacuum was maintained from a receiver.

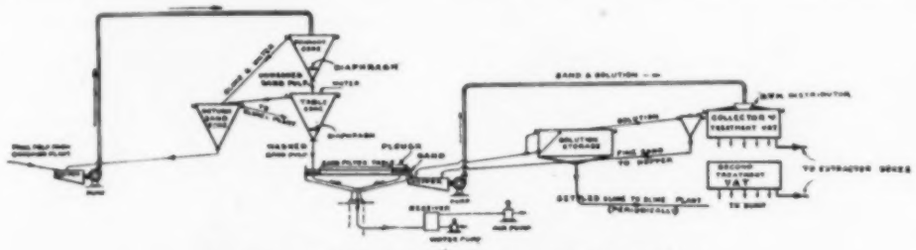


DIAGRAM OF CONTINUOUS SAND FILTRATION PROCESS.

To the lower portion of the receiver was connected a water pump and to the upper a vacuum air pump. Various other filters of the same type, but of larger capacity, have since been installed and have proven thoroughly successful in large-scale operation. Considerable details are given on the arrangement of the plant and comparative working costs. The accompanying illustration shows what is in Mr. Caldecott's opinion at present the most suitable arrangement of plant in elevation, with superimposed primary and table cones, as well as superimposed collecting and second treatment vats. The superimposed vat system under these conditions is no longer open to the objection of contaminating mill service water, while all the vats, both upper and lower, are utilized for treatment, and the cost of installing and operating transfer belts or trucks is done away with. The collecting vats used to receive the almost slime-free sand solution pulp may be of larger dimensions than usual, as settlement of slime in layers is not to be apprehended, and fewer larger units thus employed in place of more smaller ones. For the same reason the bulk of the lime used may be crushed with the ore in the battery, thus reducing the separate grinding of lime in a ball-mill or other machine to occasional periods when unusually acid ore is being delivered to the battery. Naturally the use of sand filters greatly increases the capacity of existing plants whether of superimposed or other type, by converting all the collectors into treatment vats. "The advantages which sand filters present are considerable saving in capital expenditure both for sand vats and for belt or truck transfer installation, slightly better extraction than the ordinary system unless prolonged treatment is given, a saving with superimposed vats of belt or truck transfer cost, and more slime to be treated as such at, say, 6d. per ton less than sand in vats of slightly increased diameter."

Copper.

Calculation of Blast Furnace Charges.—In the *Mining and Scientific Press* of Nov. 13, PERCY E. BARBOUR emphasizes the advantages of calculating blast-furnace charges. He describes two methods; in the first, the slag desired is assumed and the charge is calculated; in the second, the charge is assumed and the slag is calculated. The first example is based on the practice of the U. S. Smelter in Salt Lake Valley, which treats its own ores (dry and sulphide ores) together with purchased custom ores. The furnaces are 42 in. by 180 in. at the tuyeres. The conditions at the smelter require on the charge 300 lb. of ore "A," 1200 lb. of ore "B," 500 lb. of ore "C" and 300 lb. of converter slag, "D." (The assays for silica, iron, lime, sulphur and copper are given in the article.) In choosing a slag to be made, it is not the practice at this plant to make an exact singulo-silicate ($2RO \cdot SiO_2$) or other exact class slag, but to make that slag which experience has demonstrated is best suited to local conditions. This experience-dictated slag, however, approaches a singulo-silicate. Lang, in his "Matte-Smelting," considers the best slag to be generally a bisilicate of lime ($CaO \cdot SiO_2$) and a singulo-silicate of iron ($2FeO \cdot SiO_2$) in equal parts. The assumed slag in the present case is 44 SiO_2: 25 FeO = Fe 19.5; CaO 0.23. The problem is to determine the quantity of dry (silicious) ore containing high gold and silver content, and 2 per cent copper, which can be put on the charge, and the amount of lime necessary to be added. The assays of these two materials are known, and are entered on the charge-sheet. Let the required amount of silicious ore be represented by x and the amount of lime by y. The total silica on the charge is that in ores A, B, C, D (which in this case is 765 lb.), plus 69 per cent of the weight of the silicious ore, x, plus 3 per cent of the weight of the lime-rock, y; 69 and 3 being respectively the silica content of the silicious ore and the lime-rock. Therefore, the total SiO_2 on the charge is SiO_2 in pounds $= 765 + 0.69x + 0.03y$.

In like manner the total copper on the charge is the copper in the ores A, B, C, D (in this case 42 lb.), plus 2 per cent of the weight of the silicious ores. Therefore, the total copper in the charge is Cu in pounds $= 42 + 0.02x$. Further, in this special case it is calculated that 393 lb. of sulphur are in the ores of the charge, and if it is assumed that 85 per cent will be volatilized, there will be left 15 per cent for the matte, which amounts to 59 lb. In forming matte, Cu combines with one-quarter of its own weight of sulphur to form Cu_2S, because

At. wt. $Cu = 63.6$ $Cu_2 = 127.2$

At. wt. $S = 32.07$. $127.2 \div 32.07 = 4$.

Since the amount of Cu in the charge is $42 + 0.02x$, the amount of sulphur required by this amount of copper is $10 + 0.005x$, and the amount of sulphur left to combine with iron is $49 - 0.005x$. In forming the matte the sulphur combines with 1.75 times its own weight of iron to form FeS, because

At. wt. $Fe = 55.9$

At. wt. $S = 32.07$. $55.9 \div 32.07 = 1.75$.

The amount of Fe required to satisfy $(49 - 0.005x)$ lb. of S is, therefore, $86 - 0.009x$. The total Fe in the charge as per calculations, is 599 lb. from ores A, B, C, and D, plus 4 per cent of the weight of the silicious ore x, or $599 + 0.04x$. Therefore, the total Fe left to go into the slag will equal $(599 + 0.04x) - (86 - 0.009x) = 513 + 0.049x$.

The total lime (CaO) in the charge, all of which goes into the slag is 69 lb. from A, B, C and D, plus 6 per cent of the weight of the silicious ore x, plus 51 per cent of the weight of the lime-rock y. This equals $69 + 0.06x + 0.51y$. We have, therefore, three equations for the pounds of silica and iron and lime in the slag:

$$SiO_2 = 765 + 0.69x + 0.03y$$
$$Fe = 513 + 0.049x$$
$$CaO = 69 + 0.06x + 0.51y$$

In the chosen slag the ratio of SiO_2 to Fe is $44 \div 19.5$, while

the ratio of SiO_2 to CaO is 44 ÷ 23. This gives the two equations:

$$11.25\,x + 0.58\,y = 7700$$
$$13.26\,x - 21.71\,y = -14{,}570.$$

From this we find $x = 629$ and $y = 1056$, the pounds of silicious ore and limerock respectively to be added to the charge.

Hence, the charge will consist of 2300 lb. ores A, B, C, D, 629 lb. silicious ore, and 1056 lb. limerock, hence total weight 3985 lb. From the above equations we find that this charge will contain 1231 lb. SiO_2, 624 lb. Fe, 646 CaO, 55 Cu. The 55 lb. Cu on the charge will require 55 ÷ 4 = 14 lb. S for the Cu_2S in the matte. The remaining sulphur, 59 — 14 = 45 lb., will require 45 × 1.75 = 0.79 lb. Fe for the FeS of the matte. Therefore, the matte will contain 55 lb. Cu, 59 S and 79 Fe. Assuming that these elements will comprise 95 per cent of the matte, the other 5 per cent being substances not considered, there results 203 lb. of matte, equivalent to a matte fall of 203 ÷ 3985 = 5.1 per cent. The resulting matte will assay 55 ÷ 203 = 27 per cent Cu. The slag will contain all the SiO_2 in the charge, 1231 lb.; all the CaO in the charge, 646 lb.; the Fe in the charge, 624 lb., taken up by the S in the matte = 545 lb., which is equivalent to 699 lb. FeO. The slag is then composed of 1231 lb. SiO_2 = 43.9 per cent, 699 lb. FeO = 24.9 per cent, 646 lb. CaO = 23.1 per cent, which is the slag intended to be produced.

The method of calculation formerly used at the Garfield smelter is finally described. It consists in assuming the charge from past experience and the amounts which must be smelted, and calculating the resulting slag. The charge consists of 2000 lb. of bed-ore, 2500 lb. of ore B, 500 lb. silicious ore, 500 lb. slag, and 500 lb. limerock. The amounts of silica, iron, lime, sulphur, and copper are calculated and carried out on the sheet. The volatilization of sulphur is assumed at 60 per cent. There is then left for matte 167.4 lb. sulphur. Copper in the matte requires one-quarter of its weight in sulphur, or 62 lb. Thus, 167.4 — 62 = 105.4 lb. S is left to combine with the iron of the matte. Sulphur requires 1¾ or 7/4 its weight of iron to form FeS. Therefore, 105.4 × 7/4 = 164.1 lb. Fe required. This amount leaves 995 lb. to go into slag. Since Fe is 7/9 of FeO, 995 × 9/7 = 1278.9 lb. of FeO in the slag.

The slag has now been determined. It contains 1786.0 lb. SiO_2, 1278.9 lb. FeO, and 909.0 lb. CaO. Assuming that these form 92 per cent of the total slag, and that the other 8 per cent is composed of Al_2O_3, ZnO, etc., 4319 lb. of slag result; whence the above quantities give 41.1 per cent SiO_2, 29.6 FeO, 21.0 CaO. The matte will contain 2.48 lb. Cu, 164.1 Fe, 167.4 S. Assuming that these constitute 95 per cent of the total, there is a matte fall of 610 lb., equal to 10 per cent. The copper in the matte is 40.6 per cent.

The above article is critically reviewed by Prof. L. S. Austin in the *Mining and Scientific Press* of Nov. 27. He remarks that in copper smelting, quite differently from silver-lead smelting, a great variation in slag composition is permissible, and hence has risen a carelessness in the preparation of the charge. But this is no reason why not to compute a charge; however, when this has been done, the slag which is produced should be controlled by the eye as well as by analysis. Referring to the singulo-silicate slag, which Mr. Barbour proposes to use, Professor Austin says that in pyrite practice the metallurgist is chiefly anxious that his slag should carry specified quantities of silica (in this instance 44 per cent, and that the bases go as far as possible with the addition of no more dead limestone flux than needed. The total of SiO_2, FeO and CaO, adds up in the present case to 92 per cent, while in practice the sum varies between 85 and 90 for Utah ores.

Various objections are raised by Professor Austin against details of the calculation of the Garfield slag. He states that he has "given in his work of the Metallurgy of the Common Metals the charge calculations for iron, silver-lead, and copper charges based upon the trial method, which he has found practically the most used by smelter men. Since, in the final summing up, this computation is used to check the results, why not use it from the beginning?"

Zinc.

Effect of Impurities on Roasting and Distillation Process.—The various reactions which are caused by impurities in the roasting of zinc sulphide ore and in the reduction and distillation of zinc oxide are the object of a paper by V. Lepiarczyk in *Metallurgie* of July 8, 1909. The experiments were made on a rather small scale; the charges were made up not from ore, but from pure materials, and the influence of different additions was investigated in succession. In roasting pure zinc sulphide it is possible to drive off all the sulphur, if the conditions are such that any zinc sulphate which is formed during the roasting process is destroyed and SO_3 and SO_2 are carried off. If in the roasting process in practice not all the sulphur is driven off, this is due to the presence of iron (Fe_2O_3 or FeS) and limestone ($CaCO_3$). The iron is less detrimental. The presence of Fe_2O_3 manifests itself in two directions: firstly, it causes some of the ZnS to remain unchanged; secondly, there is a reaction between ZnS and Fe_2O_3 yielding FeS. The presence of lime is far more detrimental. While it does not prevent the complete change of the zinc sulphide into the oxide, yet the largest part of the $CaCO_3$ is changed into $CaSO_4$. This $CaSO_4$ not only represents a loss of sulphur, but it causes an agglomeration of the roasted blende when it is being stored so that it has to be crushed again; further, the $CaSO_4$ is reduced in the retort in the distillation process as will be shown later on. If iron is present in form of FeS together with $CaSO_4$, it prevents a small part of the zinc sulphide from being completely changed into oxide; the sulphur from the FeS itself is completely removed, but the $CaSO_4$ is not completely changed into sulphate. The results of this part of his research are summed up by the author in the statement that it is the Fe_2O_3 which prevents the complete oxidation of the ZnS, while the lime causes the formation of sulphate.

In the reduction process in the retort the reactions between the different materials which may be present were studied separately. If nothing but carbon and $CaSO_4$ are present, not the whole $CaSO_4$ is reduced to CaS, as some remains unchanged; but there are two reactions going on at the same time, namely, $CaSO_4 + 4\,C = CaS + 4\,CO$ and $3\,CaSO_4 + CaS = 4\,CaO + 4\,SO_2$. In the second experiment nothing but carbon and zinc sulphide were present; while it is generally supposed that zinc sulphide is not reduced, this was found to be erroneous; some of the zinc and sulphur evaporates and the quantities of Zn and S evaporated are in the ratio of their atomic weights. In the third experiment the reaction between carbon and FeS was studied and it was found that FeS is even more attacked than ZnS. It seems that the two reactions which take place in the second and third experiment are $2\,ZnS + C = 2\,Zn + CS_2$ and $2\,FeS + C = 2\,Fe + CS_2$. This is interesting for practical purposes since it shows that the FeS is really not so dangerous as is generally supposed because only little FeS remains back in the roasting process, and the greatest part of this is decomposed in the retort. In the fourth experiment the reaction between Fe_2O_3 and ZnS was investigated, and it seems that the complete progress of the reactions is given by the formulas $2\,Fe_2O_3 + 4\,ZnS = 4\,FeO + SO_2 + Zn + 3\,ZnS = 3\,FeO + Fe + ZnO + SO_2 + 3\,ZnS = 3\,FeS + Fe + 4\,ZnO + SO_2$. The author then studied the reactions between more than two materials. If carbon, zinc sulphide, and Fe_2O_3 are present together the reaction is $Fe_2O_3 + 2\,ZnS + 4\,C = 2\,Fe + 2\,Zn + 3\,CO + CS_2$. If carbon, zinc sulphide, and FeS are present, the reaction seems to be $2\,ZnS + 2\,FeS + C = ZnS + Zn + FeS + Fe + CS_2$. While pure $CaSO_4$, when reacting with C, yields CaS and CaO and some non-decomposed $CaSO_4$, yet when $CaSO_4$ is present together with ZnO, FeO, and MnO, there is formed only CaO besides some non-decomposed $CaSO_4$. The author concludes that the following reactions take place: (a) $CaS + ZnO = CaO + ZnS$, or (b) $CaSO_4 + ZnO = ZnSO_4 + CaO$, and, fur-

ther, $ZnSO_4 + 4C = ZnS + 4CO$, (c) $CaS + FeO = CaO + FeS$, or (d) $CaSO_4 + FeO = FeSO_4 + CaO$, and, further, $FeSO_4 + 4C = FeS + 4CO$, (e) while in presence of Mn the following additional reaction occurs, $FeS + Mn = MnS + Fe$.

The chief conclusion of the author which is of some interest for the practice is that the iron content in zinc sulphide is slightly detrimental in the roasting process because it causes a small part of the ZnS to remain unchanged. In the reduction and distillation process, however, the iron is very useful if it is present in form of Fe_2O_3 for two reasons: First, it changes any ZnS which has not been oxidized during the roasting process into oxide and thus makes it available before reduction and distillation. Secondly, when the $CaSO_4$ is reduced in the retort, the sulphur thereby liberated is caught at once by the iron oxide so that it cannot combine with zinc. If roasted blende rich in lime is to be reduced it seems useful to add some iron oxide in order to increase the yield of metallic zinc. But this would be unnecessary if it were possible to remove the lime from the ore before the roasting process, and this would be most advantageous, both for roasting and distillation. It should be tried to remove the lime by washing the ore in one process. If no calcium sulphate was present the content of Fe_2O_3 in the ore alone would be sufficient to change any remaining zinc sulphide to zinc oxide and thus permit its reduction and distillation.

Chemical Engineering.

Multiple-Effect Distillation.—At the recent Philadelphia meeting of the American Institute of Chemical Engineers two papers were presented on multiple-effect distillation. In a paper by F. J. WOOD (mechanical engineer of Marx & Rawolle's glycerine refinery), the various steps leading up to the present development of the art of glycerine distillation are described with particular reference to the fuel economy. The consumption of coal per pound of glycerine output is greater now than with the earlier system, as it has been found more economical to save glycerine from decomposition by the use of a flow of steam in a vacuum which greatly lowers the boiling point. In the multiple-effect apparatus, a superior economy of fuel is obtained by using but one flow of steam to a number of stills and condensers, the stills being arranged in series in regard to the steam flow—that is, the steam from the steam jet of the first still flows through a series of stills alternating with condensers, the condensers being maintained at a temperature that will condense glycerine out of the steam, but not the steam itself. The glycerine is forced through heaters by means of centrifugal pumps. The heaters are connected with the boilers and supply the heat necessary for the vaporization of the glycerine. From the heaters the hot glycerine is sprayed down through the steam.

The combined vapors of steam and glycerine pass into boiling water condensers whose temperature is maintained at such a point by means of a pressure regulator, that the glycerine, but not the steam, is condensed. The glycerine in condensing gives up its latent heat of evaporation to the boiling water. The steam passes on to the next still in the series to take up its quota of glycerine. Finally, after the last of the series of stills is passed, the steam, together with the volatile impurities, are condensed as sweet water in a cold water condenser. In order to maintain a uniform vacuum throughout the system, there must be a free flow of steam and glycerine vapor without obstruction. In this system, instead of forcing the steam down through the body of glycerine, the glycerine is pumped up and sprayed down through the steam and no back pressure to the flow of steam results.

The paper by A. C. LANGMUIR, also presented before the American Institute of Chemical Engineers, deals with the general advantages and applications of multiple-effect distillation. In the author's opinion the wood process of glycerine distillation marks one of the most important advances made in recent years in this line. The patent rights have been acquired by Marx & Rawolle, New York, in whose plant the process is now in successful use on a large scale. (See abstract above.) The fuel economy is marked, a considerable increase in output per still being obtained at a fuel consumption of less than half of that formerly used. A set of six stills in series is successfully at work with no diminution of efficiency in the last still as compared with the first.

The chief advantages of the process, aside from the fuel economy, consist in the ease of working, uniformity of product with varying crudes and the scientific control possible at all times. The process is largely automatic, less labor and less skill in handling being required. Only a small body of glycerine is present in the heaters, stills and connections, and the prolonged heating of large bodies of glycerine with resulting loss in the average still is avoided. The spraying of the glycerine through the steam avoids the shooting and foaming often met with when distilling certain crudes in the still generally used. The process is applicable in any distillation where steam is used and the product obtained has a boiling point higher than that of water, such as the distillation of the heavier products of coal tar and petroleum, of aniline, essential oils and the fatty acids of the candle industry.

Smokeless Combustion and Boiler Efficiency.—At the recent Philadelphia meeting of the American Institute of Chemical Engineers a paper was presented by A. BEMENT, Chicago, Ill., giving the results of tests carried out to ascertain the loss of heat due to incomplete combustion. The results are given in a table showing the comparative heat balance of an ordinary water-tube boiler in which the combustion was incomplete and an improved boiler arranged for the reception of a fire-brick tile furnace roof which made it necessary for the gases to travel to the rear of the boiler before coming in contact with the water tubes. By this arrangement, complete combustion was secured, as shown by the absence of smoke. The loss of heat due to incomplete combustion in the ordinary boiler was found to be 9.3 per cent, while in the improved boiler there was no loss. The method of making the determinations and calculations was given in full. In the second part of the paper, the causes governing the efficiency of a boiler were discussed, on the basis of the results of the efficiency of three boilers. The first was a well-known water-tube boiler in which the gases entered among the tubes at the rear and passed diagonally across the tubes to the exit. In the second boiler, a baffle was inserted among the tubes so that gases flowed twice the length of the tubes. In the third boiler, two baffles were inserted so that the gases flowed three times the length of the tubes. The fuel saving in the single-baffled over the plain boiler was 8.68 per cent, while the saving in the double-baffled was 20.68 per cent. Complete data with reference to the coal, temperature of gases, steam, draft, power development, etc., were given.

Oil and Grease Extraction.—In a paper read at the Philadelphia meeting of the American Institute of Chemical Engineers, Mr. W. M. BOOTH discussed the properties of the solvents which can be used for the extraction of oils, giving the solvent power and the cost of each. On this basis the cheapest solvent is found to be naphtha. The construction of building tanks and extraction apparatus was discussed. Methods of operation were given, also an account of the methods in use for the recovery of grease from garbage tankage and other waste products is given; drawings of extraction apparatus, and a number of tables and curves giving the properties of various fats and oils.

Utilization of Waste India Rubber.—In a paper read at the Philadelphia meeting of the American Institute of Chemical Engineers, Mr. S. P. SHARPLES gave a résumé of the various methods which have been employed in recovering the waste rubber of commerce, calling particular attention to the processes of the late Nathaniel Chapman Mitchell in the Philadelphia Rubber Works. Mr. Mitchell was the first to make a commercial success in this country of methods for utilizing worn-out rubber articles, such as boots and shoes. This was largely due to the fact that he discovered that sulphuric acid of 15° B.

to 25° B. could be employed to remove the fiber from these goods without injuring the rubber. The process was at once stolen and utilized all over the country until at the present time it is used in many works.

Glass and Iron.—Although the formation of a special Chemical Reichsanstalt in Germany is contemplated some chemical research work is carried out at present at the Physikalisch-Technische Reichsanstalt in Berlin. From the report of the work done in 1908 given in London *Engineering* of Dec. 17, we notice briefly some of the more interesting chemical investigations: They concern first the weathering of crown glass by means of the cosin reaction; the glasses must not be dipped into the solution for much more than a minute, lest the lead be attacked in addition to the alkali. The firm of C. A. F. Kahlbaum has purified large bulks of iron nitrate, $Fe(NO_3)_3 + 9\,H_2O$, which is melted at 50 deg. Cent. and exposed to gaseous ammonia; dense red iron oxide and ammonium nitrate result. The latter is washed out and the oxide reduced and the powdery metal melted in the oxy-hydrogen flame. An iron rod thus prepared contained 0.001 per cent of manganese, 0.004 per cent of copper, the same amount of nickel or cobalt, traces of carbon and silicon, and a few hundredths per cent of oxygen. Reduced oxalate, also from Kahlbaum, was less pure. The physical tests of this "nitrate" iron and of other iron gave the following results:

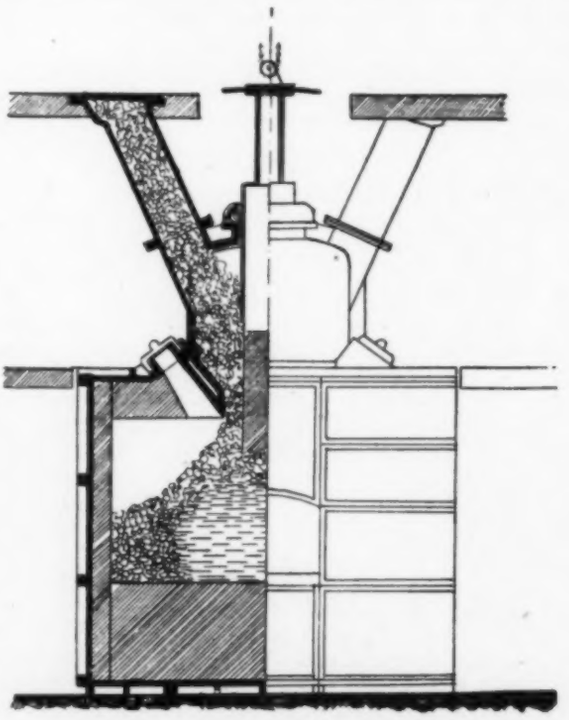
	Nitrate iron.	Kreusler wire.	Iron 0.1 per cent of carbon.	Steel 1 per cent of carbon.
Electric conductivity at 18 deg. Cent.	10	9.6	8.3	5×10^{-4}
Temperature coefficient of this conductivity	5.9	5.5	4.5	3.7×10^{-3}
Magnetic coercitive force	1.4	2.6		

This nitrate iron, which is superior to most electrolytic iron, will be valued by chemists for preparing their titration solutions and by physicists, but it is not a "pure" iron yet, of course.

Electrochemistry.

Large Electric Furnaces.—The tendency in electric furnace construction is toward a rapid increase of the size. This is clearly shown in the present development of the electric steel furnace and it is well known that the chief difficulty encountered in increasing the size is found in the construction of the electrodes. In the *Zeit. f. Electrochemie* of Aug. 1 R. Taussig describes the design of A. Helfenstein for large furnaces for calcium carbide manufacture. The size is limited by the amount of power which a single electrode or a compact electrode bundle can carry into the furnace. According to the results obtained in actual practice the maximum so far obtained is 2500 or 3000 kw per electrode (30,000 to 40,000 amp at 75 to 90 volts). For a three-phase furnace with three electrodes this means a total capacity of 7500 to 9000 kw or 10,000 to 12,000 hp. A further step has been made in the design of a double three-phase furnace in which six instead of three electrode bundles are placed in the same furnace shaft. This means two separate three-phase units in the same shaft, so that the capacity of such a furnace is 15,000 to 18,000 kw corresponding to a production of 80 to 110 tons of carbide in 24 hours in one furnace. With such large furnaces the heating effect of the reaction gases of the furnace becomes very great and must be taken care of in order to protect the workingmen and the connections to the external circuit. Further the smoke becomes troublesome (with ferro-silicon even more than with carbide). The greatest difficulty is, however, found in the method of charging the furnaces so as to arrange the charge properly around the electrode. It seems advisable with large furnaces to build them air-tight so that the smoke trouble is overcome and the heat of the gases of the reaction can be usefully employed. The adjoining illustration shows a vertical section of a single electric hearth for 8000 to 10,000 hp. About 8 or 10 meters above the bottom of the furnace a special charging floor is arranged. At the top of the furnace there is placed a large and tightly closed mixing receptacle or hopper for the charge, through the center of which the electrode passes down (its weight is 3000 to 14,000 kg for 10,000 hp). During the operation the electrode is surrounded on all sides by the charge. The charging receptacle has a capacity of 5000 to 7000 kilograms and is connected by means of long pipes with the charging floor. The charge is continually supplied through these pipes into the charging receptacle and passes from there into the electric furnace. In order to control the exact supply of the charge into the furnace, but especially for the removal of any furnace troubles, there are provided in the furnace at the side of the hopper broad slits which can be tightly closed. For a large three-phase furnace of 25,000 hp three separate electrode bundles are used. Each electrode corresponding to one phase is provided with its own charging arrangement. It does not seem, however, that this design of large furnaces has already been tried out in practice.

Silundum.—At the last meeting of the German Bunsen Society Dr. R. Amberg read a paper on the manufacture of silundum in a carborundum factory in Bodeo, Switzerland. The



LARGE CALCIUM CARBIDE FURNACE.

paper is printed in *Zeit. f. Electrochemie*, October 1. They prepare amorphous carbon articles by pressing or by other treatment so that they get the desired form and then heat them in electric resistance furnaces of the simplest type so as to subject them to the action of carborundum and sand, or carbon and sand, or to both actions together. The two reactions are $SiO_2 + 2\,SiC = 3\,Si + 2\,CO$ and $SiO_2 + 2\,C = Si + 2\,CO$. The silicon permeates into the carbon article and forms silundum. (Chemically this is the same material as what is called siloxicon by Dr. Acheson.) As has been mentioned in our columns, silundum is useful as an electric conductor at high temperatures, for instance, for electric heating apparatus, etc. The resistivity of silundum is about six times that of carbon. The temperature coefficient of the resistivity is negative, but very small. At 1000 deg. Cent. silundum, according to its constitution, has a resistivity of one-half or two-thirds that at ordinary temperature. Silundum articles can stand temperatures of 1600 deg. for a long time without crystallizing and for short periods even temperatures up to 1700 deg. Above 1700 deg. silicon evaporates, yielding a white vapor of SiO_2, while the carbon remains back as graphite. Silundum can be heated to white heat and then immediately put into cold water without

cracking. It is also proposed to treat coke in the same way and to use the coke which has been changed into silundum instead of ferro-silicon in steel works. Silundum has been used as an electrode in electrolytic processes, but experiments on its use as a unit in sodium chloride electrolysis have not yet been concluded. Air attacks silundum even at the highest temperatures only in so far as a film of SiO_2 is formed. This protects, then, the internal parts.

Calcium Cyanamide.—It is well known that the formation of calcium cyanamide from calcium carbide and atmospheric nitrogen (which must first be separated from the oxygen) requires a temperature of at least 1000 to 1100 deg. Cent. in order to change all the carbide into cyanamide. It is true that the same reaction goes on at a lower temperature, like 800 or 900 deg. Cent., but it soon stops and it stops the earlier the lower the temperature. However, if a few per cents of calcium chloride are added to the carbide, a complete transformation of the carbide into the cyanide can be obtained at a lower temperature, like 700 or 800 deg. (process of Polzenius, German patent 163,320). Also, if some calcium fluoride is added to the carbide (process of F. Carlson) it is possible to obtain a product rich in nitrogen at 900 deg. Cent., while at 800 deg. Cent. the fluoride shows its favorable effect, but does not change all the carbide into cyanamide. A paper by F. FOERSTER and HANS JACOBY in *Zeit f. Electrochemie*, Nov. 1, is quite interesting as it shows that the action of the calcium fluoride addition is essentially different from that of the calcium chloride addition. If the quantities of calcium fluoride which are added are varied it makes practically no difference if only the heating process is carried on for a long-enough time. Whatever the amount of calcium fluoride added, the absorption of the nitrogen by the calcium carbide remains limited at 800 to 860 deg. The amount changed to cyanamide is the same whether 1, 2, 5 or 10 per cent of fluoride are added. The amount of the addition of fluoride determines only the speed of the transformation of the carbine into cyanamide. The formation of calcium cyanamide evolves considerable heat, but it is remarkable that the temperature attained by the charge containing calcium fluoride is less than that of charges without any additions or that of charges to which calcium chloride is added. In the latter cases the temperature may rapidly rise to 1000 deg. Cent. If, however, an addition of calcium fluoride is used, the temperature remains near 900 deg. Cent. The addition of calcium fluoride has, therefore, not only the advantage that it reduces the reaction temperature by 100 deg. Cent., but it permits a more quiet and more uniformed reaction. This is of importance when the retorts in which the process is carried out are heated from the outside. This has formerly been tried since in view of the relatively low temperature required electric heating was thought to be unnecessary. But as was described in our articles on the large new calcium cyanamide works in Odda, Norway, (our Vol. 7, pages 212, 309 and 360), it has been found preferable to use relatively small units and to heat them from the inside by electric resistors. Under these circumstances it is not necessary to use any addition in order to reduce the temperature of the reaction. But the present authors think that the addition of calcium fluoride might have the advantage of providing greater uniformity and regularity of the process. It is known that when nitrogen reacts with carbide (especially barium carbide) not only cyanamide but cyanide is formed. In this respect the addition of calcium chloride has again a different effect from the addition of calcium fluoride. With the fluoride some cyanide is obtained, with the chloride none.

Nickel Wire as Resistor for Laboratory Furnaces.—A paper by Prof. M. LE BLANC, read at the last meeting of the German Bunsen Society and printed in the *Zeit f. Electrochemie*, Sept. 15, gives details of an electric laboratory furnace in which nickel wire is used as a substitute for platinum. The results have been very satisfactory. In the discussion several speakers (among them Prof. Nernst) stated that they had obtained good results with nickel, but others complained of the great fragility of nickel. (The nickel-chrome alloy wire, made and used quite extensively in this country, would probably prove far more satisfactory for such work than pure nickel; it does not yet seem to have been introduced in Germany.)

Electrolysis of Alkali Chlorides.—A sketch of the development of the mercury cathode process in Europe in four successive types is given by R. TAUSSIG in *Zeit. f. Electrochemie*, Sept. 1. The original type—the old Castner-Kellner cell—was rocked. The cell had the great disadvantage that its units were too small, the current used for each cell being only 400 or 500 amp. The electrolytic plant with all accessories became therefore very large and somewhat complicated and the installation and attendance was expensive. The second type is the Kellner cell, which consists of three compartments, the chlorine compartment being in the center. The three compartments are connected together by siphons and pressure can be applied on the mercury in the two outer compartments by means of compressed air so as to force the mercury from one outer compartment into the middle compartment and then into the other outer compartment. When passing through the middle compartment the mercury alloys with the sodium which is being set free there. The anodes are platinum nettings, size 50 by 50 mm. They are arranged in electrode boxes of concrete which have proved more successful than the old clay boxes. Each box contains 88 platinum nettings so that each chlorine cell contains 528 platinum nettings of 525 grams weight, their capacity being 4000 amp. In the two outside compartments cathodes of cast-iron grids are placed. The third type is the Kellner-Solvay cell, which is now in use in Jemeuppe in Belgium, Brescia in Italy, and Lubimof in Russia. The mercury is circulated with the aid of a wheel. This cell represents the largest unit. It was originally built for 6000 amp, but is operated now at 10,000 amp and the current may be increased even to 15,000 amp. The fourth type is a modification of the latter cell and the only difference is that the mercury is circulated in it not by means of a wheel but by by means of an Archimedean screw.

Electrolytic Bleaching.—The electrolytic bleaching process as used in this country employs essentially electrolysis of sodium chloride for the production of caustic soda and chlorine, the latter being changed to bleaching powder; bleaching solutions are then made from the bleaching powder. On the other hand, in Europe the direct production of bleaching hypochlorite solutions directly in the cell in which the sodium chloride is electrolyzed has assumed considerable proportions in practice. On the basis of considerable experience it is claimed that the directly produced hypochlorite solution has a better bleaching effect on cotton and textile fabrics than solutions made from bleaching powder. (A hypochlorite cell of this type is made and used in this country for laundry work.) On the other hand, there has been a difference of opinion concerning the bleaching of sulphite pulp. In a paper by B. FRASS in the *Papier Fabrikant*, Vol. 7, No. 10 and 11 (also in *Zeit. f. Electrochemie*, June 15) he reports on an investigation from which he concludes that in sulphite cellulose bleaching, too, the directly produced hypochlorite solution is more effective than a bleaching powder solution. The behavior is as though the active chlorine in form of the hypochlorite solution was more effective than in form of a bleaching powder solution. An exact explanation has not yet been found.

Formation of Storage Battery Plates.—G. JUST, P. ARKENASY and B. MITROFANOFF gave in *Zeit. f. Electrochemie*, Nov. 15, an interesting account of extended researches on the use of an addition of nitric acid or nitrates to sulphuric acid electrolytes for the rapid formation of positive lead accumulator plates. The formation of plane lead plates was first studied in pure sulphuric acid according to the process of Plante; the development of capacity was investigated and charge and discharge curves were plotted. Analogous experiments were then made with an addition of nitrate to the sulphuric acid. They

showed that the capacity could be quickly increased in this way and also showed a change of form of the charge and discharge curves. The authors endeavored to find the best amount of nitrate to be added, but had some difficulty because the active mass had a tendency to drop off from the plane lead plates. Nevertheless, the experiments showed that approximately a concentration of 30 grams of potassium nitrate per 218 grams of sulphuric acid per liter was specially favorable. If too much nitrate is added the tendency of the active mass to drop off is increased. The less the addition of nitrate the more adherent becomes the active mass. They also found that when nitrate is added a white lead sulphate film is first formed on the positive plates. All attempts to improve the adhesion of the active mass to the plane lead surfaces were successful. But it was found that when grids such as used in practice with a large surface (Grossoberflächenplatten) were treated in the same way with the addition of nitrate, it was possible to get the formation of an active material which would be sufficiently adhesive in practice. The more nitrate is added, the better is the porosity of the active mass formed, while the durability is diminished. Since both effects act against each other in practice, there must be a certain concentration of nitrate which is most suitable.

Recent Metallurgical and Chemical Patents.

Iron and Steel.

Removal of Phosphorus.—Probably one of the most interesting advances made in recent years in steel refining is disclosed in a patent by Mr. Ernest Humbert (Dr. Héroult's metallurgist who is at present operating the Héroult electric furnace plant of the U. S. Steel Corporation in South Chicago). The invention consists of removing the phosphorus, not as usual, as phosphate, but as phosphide. The claims made for the process are that there is less loss of time and labor than in the old process, that there is an increase in the amount of steel produced, and that there is a saving in the electrical energy required for the electric furnace. The invention consists in the use of an oxidizing slag in such a way as to combine with the phosphorus and form a phosphate, and the adding of a reducing material to the slag containing the phosphate to convert the latter into a phosphide. The reducing material, usually ground coke, floats on top of the slag and reduces the phosphate without interfering with the metal below. "The strength of combination of the phosphide is so great that the phosphorus cannot be separated out of it by the iron without its first being changed back to phosphate, and this cannot occur in a reducing atmosphere." The fact that the reducing atmosphere is required seems to necessitate the use of an electric furnace. The following example of the operation of the process in practice is given: An impure charge of steel from a Bessemer converter was used, analyzing approximately phosphorus 0.10 per cent, sulphur 0.16, manganese 0.10, silicon traces, carbon 0.07. This was transferred to a Héroult electric furnace. A black slag containing 400 lb. of mill scale and 400 lb. of lime was added to a 15-ton charge of steel, making an oxidizing slag, which became fluid when the steel below was thoroughly oxidized and all the phosphorus had passed into the slag as phosphate of lime P_2O_3 (CaO). Upon this slag containing the phosphate of lime was added ground coke, which floated thereon and reduced the phosphate therein, forming calcium phosphide (P_2Ca_3). Thereupon, without removing the slag, the desired additions (silicon, manganese, etc.) were made to the molten steel. There was no return of phosphorus to the steel, and the final product contained phosphorus 0.005 per cent, sulphur 0.005, with carbon from 0.05 to 1.50 and silicon and manganese ad libitum. (943,192, Dec. 14, 1909.)

Electric Steel Furnace.—As was noticed in our June issue, 1909 (vol. 7, page 255), Mr. C. A. Keller has abandoned the type of electric steel furnace formerly employed by him, in which the current passes from one electrode above the bath into the bath and out of it again into another electrode above the bath (the well-known Héroult type). He now employs, like Mr. Girod, only one electrode at the top, or if he uses several electrodes they are all connected in parallel; the other terminal is the bottom or the hearth of the furnace. The current, therefore, passes from the electrodes at the top into the bath and leaves the furnace at the bottom. For this type a durable construction of the furnace bottom is, of course, all important. In a recent patent of Mr. Keller a bundle of iron bars is placed vertically in the furnace bottom and are all connected electrically to a conducting plate arranged below their bottom ends. Between these bars and around the bundle which they form is strongly rammed in a refractory earth, preferably magnesia earth. The furnace bottom consists, therefore, half of vertical iron rods and half of magnesia earth. The latter also becomes a conductor at high temperatures. (941,419, Nov. 30, 1909.)

Electric Reduction Furnace for Iron Ore.—The Swedish electric reduction furnace for iron ore which has been described repeatedly in this journal (see especially the article by Yngström, page 11 of this issue) is patented by E. A. A. Grönwall, A. R. Lindblad and O. Stalhane. It will be sufficient here to quote the first claim, which relates to "the combination in an electric furnace with the heating chamber and the stack communicating therewith, the chamber being organized to provide a free space between the charge and the top of the heating chamber when the charge is introduced into the chamber and into the lower portion of the stack, of electrodes entering through the wall of the chamber and extended to pass through the said free space with their end portions immersed in the charge, and means for drawing gas from the upper part of the furnace and passing such gas into said free space portion of the furnace. (943,224, Dec. 14, 1909.)

Copper.

Converter Construction.—In customary practice of bessemerizing copper matte, a silica lining is used; it has the disadvantage that it unites with the iron present to form a slag, so that the lining must be frequently renewed. If a noncorrodible basic lining is used, made of magnesite or chrome brick, etc., there results a considerable expansion under the influence of the high heat, and this expansion continues practically throughout the life of the lining, which should last for several months. The usual result is either disintegration of the lining or rupture of the converter shell. To overcome this difficulty, Mr. Wm. H. Peirce (Baltimore Copper Company) proposes to arrange the basic lining so that it is free to shift relatively to the shell, so as to expand under the high heat developed in the course of treatment and at the same time preserve its continuity, while the twyers embedded in the lining have full delivery connection with the blast supply. In the first arrangement employing this scheme the twyers were flexibly sustained to shift with the lining to thereby avoid the breaking down of the twyers or the lining. Leaks through the lining, which would tend to disintegrate it, were thereby avoided. This arrangement has now been changed by Mr. Peirce in so far as the lining is shiftable as before relatively to the shell, but the section of the lining in the immediate neighborhood of the twyers is anchored to the shell to form a zone in which there is no substantial shift. The cylindrical horizontal shell is open at its upper side to permit expansion of the lining, and is provided with a tenacious, coherent, noncorrodible lining arched over at the open top of the shell and free to expand therein, and the set of blast twyers, extending

through the lining, is arranged longitudinally at one side of the shell. In such a converter the expansion of the lining is chiefly in a vertical direction or transerve to the axis of the shell. To prevent this shift in the immediate neighborhood of the twyers the section of the lining at one side of the converter and at the longitudinally extending twyer belt is anchored to the shell, the lining being otherwise free to expand in opposite direction from the twyer belt. The twyers are embedded in the lining and an abutment, or a series of abutments of metal, secured to the shell and extending into the lining, form a longitudinally extending shelf or barrier adjacent the set of twyers. (942,661, Dec. 7, 1909.)

Zinc.

Electric Zinc Smelting.—Mr. F. T. Snyder, who has already patented quite a number of different designs for electric zinc furnaces, has recently patented another one operated by three-phase currents. The furnace is of a narrow oblong shape and the three electrodes suspended from above are arranged in a row so that the side walls may be close to the source of heat. The hearth of the furnace is filled during operation with molten lead upon which rest the molten matte and slag. The furnace is open at the top to introduce the charge and permit the escape of the carbon monoxide gas which is burned at the top. The furnace is water-jacketed up to the top of the furnace to cool the upper portion of the charge. Powdered coke is introduced through a hopper surrounding the top of the electrode, while the main body of the charge is put into the furnace between the outer edges of this hopper and the water-cooled sides of the furnace. The zinc vapor rises in this outer part, but condenses on the water-jacketed walls "in liquid form rather than as dust." The condensed liquid zinc is withdrawn through drains consisting of thick hollow tubes of graphite, inserted in the walls and provided with a special water-jacket. The bores of these tubes are small so that they are completely filled by a small stream of liquid zinc. The zinc drops off at the outside into a well, which surrounds the lower part of the furnace and is at the bottom in connection with a molten lead receptacle in the hearth of the furnace. The advantage is that any of the heavier metals such as lead which may have been volatilized with zinc will settle in the well and join the main body of the lead in the bottom of the crucible. (942,110, Dec. 7, 1909.)

Deposition from Zinc Sulphate Solution.—In depositing zinc from sulphate solution by electrolysis, the electrolyte becomes gradually richer in free sulphuric acid and this results in part of the zinc deposit being redissolved from the cathode. Messrs. V. Engelhardt and Max Huth have found that this is due to impurities in the electrolyte (for instance, metal dissolved from the anode). They, therefore, employ a pure solution of zinc sulphate and an anode from which nothing is dissolved during electrolysis. This is pure massive or compact lead peroxide on a non-metallic support. If every other metal and even carbon be completely excluded the content of free sulphuric acid may be allowed to exceed 10 per cent without conversion or dissolution of the zinc already deposited. The patent is assigned to Siemens & Halske Company. (935,250, Sept. 28, 1909.)

Precious Metals.

Chlorination Process.—For the chlorination of refractory ores Mr. Wm. Koehler proposes to subject them to the action of hydrochloric acid in the presence of hydrogen at an elevated temperature, which must be below the temperature necessary for the volatilization of the metallic chlorides formed. The process is preferably carried out in a tube mill, hydrochloric acid entering together with the hydrogen. For commercial purposes very good results are obtained by using hydrochloric acid together with water gas, in which case the acid is acting in the presence of hydrogen and a carbon compound. (942,569, Dec. 7, 1909.)

Aluminium

Welding of Aluminium.—The Schoop process of aluminium welding by means of a special flux which simply serves to remove from the surface the otherwise present oxide film, but does not introduce any foreign materials into the weld, has been described in various articles in former volumes of this journal. A new improvement in the constitution of this flux is described in a recent patent of Mr. M. U. Schoop. It is a mixture of alkali chlorides with fluorides; for instance, potassium chloride 60 grams, sodium chloride 12 grams, lithium chloride 20 grams, cryolithe 6 grams. The materials are melted together and the molten mass is collected and ground to powder and then mixed to a thick paste with an aqueous solution of glycerine containing 5 to 10 per cent of glycerine. The proportions of the constituents of the welding compound are so chosen that the compound fuses at a temperature close to that of the melting point of aluminium. (943,164, Dec. 14, 1909.)

Light Metal.—Mr. J. G. Mellen patents a metal, consisting of aluminium 79 parts, magnesium 11 parts, phosphorized copper 10 parts. By phosphorized copper he means a metal consisting of 99.5 per cent copper and 0.5 per cent phosphorous. The metal, which he calls "duralum," is said to be adapted for general use as an anti-friction metal for bearings, etc., and for use in castings for motor cars, etc. It has approximately the same specific gravity as aluminium and "is very much harder and stronger, it being particularly adapted to resist great crushing strain. For the purposes for which light castings are required it is much superior to iron, as it will stand even greater compression and does not dent under blows, and while fully as light as aluminium it is wholly free from the objectionable qualities of aluminium." The metal is said to run freely in casting and to machine well. (936,339, Oct. 12, 1909.)

Aluminium Solder.—An aluminium solder, patented by Mr. G. Hartmann, consists of 80 per cent tin, 17 per cent aluminium, 0.7 per cent nickel and 2.3 per cent metallic magnesium. (938,422, Oct. 26, 1909.)

Another aluminium solder, patented by Mr. L. Goppman, consists of 49.05 per cent tin, 3.43 antimony, 26.09 lead, 20.31 zinc and 1.10 copper. (939,494, Nov. 9, 1909.)

Tin.

Detinning.—A new chlorine detinning process is referred to in a patent of Messrs. W. W. Murray and H. M. Fernberger, in which tin scrap is detinned by means of chlorine gas dissolved in carbon tetrachloride. The resulting stannic chloride also dissolves in the carbon tetrachloride. After the detinning process is completed the solvent containing the stannic chloride is withdrawn from the scrap. But a considerable percentage will be retained in the numerous interstices of the scrap, which is therefore washed again with fresh carbon tetrachloride. The stannic chloride may be separated from the wash liquid by water and the solvent recovered, an aqueous solution of stannic chloride resulting from the treatment of the wash liquid with water. After the wash liquid has been removed from the residues, the portion adhering to the residue may be removed by further washing the residue with water and by the application of heat. (943,986, Dec. 21, 1909.)

Tin from Tin-Lead Waste.—Mr. A. Nodon recovers the tin from tinned lead-plate waste, etc., by subjecting the metal waste to the action of a hot solution (40° C.) of stannic sulphate with the addition of sulphuric acid, mixed with an alkaline stannic haloid salt (ammoniacal bichloride of tin), which solution has no chemical action on lead. The tin is dissolved in the form of ammoniacal chloride of tin. The solution is electrolyzed between graphite anodes and pure tin cathodes with a cathodic current density of 1 amp to 2 amp per square decimeter. (940,471, Nov. 16, 1909.)

Electroplating.

Nickel Plating.—Dr. E. F. Kern, who has done considerable work on the theory and practice of electroplating, has invented a process of depositing nickel. A smooth, dense, coherent and adherent deposit of nickel may be obtained by the

electrolysis of a bath containing fluosilicate of nickel in solution, and particularly good results may be obtained by the electrolysis of a bath containing in addition to the fluosilicate of nickel either an alkaline fluoride, as ammonium fluoride, alone, or an alkaline fluoride and a soluble fluosilicate which is more basic than nickel, preferably aluminium fluosilicate. The electrolyte recommended by him is made as follows:

Water	100	or more	parts
Fluosilicate of nickel	10	"	"
Ammonium fluoride	5	"	"
Aluminium fluosilicate	5	"	"

Ammonium fluoride should be added to the bath from time to time during the operation of the process, in order to restrain the separation of the silica. (942,729, Dec. 7, 1909.)

Galvanizing Barrel.—To reduce the cost of electrogalvanizing by making the operation almost entirely automatic is the object of a patent granted to L. Potthoff, whose work along these lines has been described in an illustrated article in our August issue, 1907 (Vol. V, page 331). The arrangement contains a revolving barrel lined inside with cocoa-matting, in which the articles to be plated are constantly tumbling over each other, so that they are plated on all sides. When a sufficient coating of material has been formed, the direction of rotation of the barrel is reversed, whereby by an automatic device the plated articles are discharged from the barrel and passed through a chute into the washing drum. Upon further charging of the plating barrel with another lot of articles, and when the barrel is now being again rotated in the opposite direction, the articles collected in the washing drum are automatically discharged and carried successively through the draining drum, the sorting drums, and finally to the receptacles for shipment or storage. The operations of galvanizing, washing, drying and sorting (various shapes and sizes of articles being galvanized at the same time) are therefore automatic, and since a single attendant can manage a number of electroplating machines, the cost of attendance is greatly reduced. (941,535, Nov. 30, 1909.)

Nitric Acid.

Fixation of Atmospheric Nitrogen.—In the October issue, page 430, we gave an illustrated description of the Innsbruck plant where the Pauling process for the fixation of atmospheric nitrogen is in commercial operation. In this connection two recent patents of Harry Pauling are interesting. Since the process was described in detail in the article referred to, we simply give here the claims of the two patents. Patent 939,441 refers to "apparatus for producing electric discharges having in combination stationary electrodes with a distance between them large enough to prevent ignition, a conductor successively passing, and thereby alternately increasing and decreasing the distance between the said electrodes and blast pipes adapted to extinguish the discharges before short-circuiting between the electrodes may occur." Patent 939,442 refers to the "process consisting in elongating the voltaic arcs produced at the place of greatest proximity of the electrodes by causing them by suitable means to propagate along the electrodes and in passing the gas to be acted upon by electrical discharges in currents of equal quantity and velocity in inclined direction against each other, and meeting in the plane of the electrodes." (939,441 and 939,442, Nov. 9, 1909.)

A patent granted to Prof. Fritz Haber and A. Koenig, of Karlsruhe, and assigned to the Badische Company, is quite interesting because it digresses for good reason from orthodox principles. The chief features of their new process are the use of a reduced pressure and the intentional cooling of the arc. Lord Rayleigh had formerly investigated the effect of a change of pressure, but had found the yield to be almost independent of the pressure. The present authors have confirmed this if nothing but the pressure is changed, and if as high a temperature as possible is employed. They found that it is different, however, if the arc is cooled by artificial means. The reduced pressure assists this cooling since the discharge which at ordinary pressure possesses the shape of a thin rod, expands and approaches the walls of the apparatus, which, being cooled, help to remove the heat from the arc. In this way discharge flames can be produced which practically no longer possess the well-known brown zone in which the nitric oxide is believed to decompose partially back again to nitrogen and oxygen. They use metallic electrodes or electrodes of magnesium oxide, calcium oxide, etc. The pressure is maintained at not more than half an atmosphere, while the apparatus is of such a shape that the inside is practically filled with arc flame, while the apparatus is cooled either by the natural escape of heat or by additional means of cooling. By this means the concentration of the nitric oxide in the gases leaving the apparatus is increased "and it is easy to produce gases containing from 9½ to 10 per cent nitric oxide when using air, and from 13½ to 14 per cent when a mixture of equal quantities of oxygen and nitrogen is employed." These yields of nitric oxide are obtained with it being necessary to cool the gases rapidly by removing them quickly from the sphere of reaction, for instance, by withdrawing them through metallic coolers. For instance, the aforementioned yields were obtained when the arc was burned inside a tube of quartz which was cooled with water and the air or mixture of oxygen and nitrogen entered and left the tube at a pressure of 90 mm mercury, and with the low velocity of 1/5 m per second. In order to prevent decomposition of the nitric oxide at the red-hot electrodes, the entry and exit tubes for the gases can be so placed that the layer of gases around the electrodes is not renewed. The sectional current density, i. e., the strength of the current per unit area of the cross-section of the tube, should preferably be as high as possible. There are no sharp limits within which the pressure should be maintained. But no advantage is gained by reducing the pressure considerably below 80 mm mercury. In the first three claims the pressure is defined as "not exceeding that of half an atmosphere." (938,316, Oct. 26.)

Alloys.

Ferro-Titanium.—If ferro-titanium is produced with carbon as reducing agent the resulting alloy is rich in carbon, which is a clear disadvantage. If, however, silica is present, the silicon reduced simultaneously with the titanium acts as a carbon excluding agent, and the alloy produced is low in carbon, but high in silicon. The silicon may be removed, however, by a following reduction with iron oxide; for instance, hematite ore. Mr. F. M. Becket (Electro Metallurgical Company) proceeds, therefore, in the following way: He produces a low-carbon high-silicon ferro-titanium in an electric furnace from a charge of titanium oxide and silica, carbon, and iron or iron ore. To the high-silicon alloy thus produced is then added iron oxide, either in the same furnace or in a separate vessel, to effect the oxidation of silicon in the alloy with simultaneous reduction of the iron oxide. A low-carbon and low-silicon ferro-titanium is thereby obtained. For instance, a product containing about 30 per cent of titanium and 2 per cent of carbon, substantially free from silicon, may be made by reacting with ferric oxide on an initial alloy containing about 50 per cent of titanium, 40 per cent of silicon, 3 per cent of carbon and 7 per cent of iron. A product containing 30 per cent of titanium and 5 per cent of carbon, substantially free from silicon, may be similarly made from an alloy containing 41 per cent of titanium, 22 per cent of silicon, 7 per cent of carbon and 30 per cent of iron. (941,553, Nov. 30, 1909.)

Calcium Nitride.—A modification of the well-known Acker process is the subject of a patent of Mr. C. E. Acker. Instead of acting on the fused lead-sodium or lead-calcium alloy with steam (in order to produce the hydroxide) he acts on it with nitrogen and produces a nitride. The nitride is usually lighter than the lead and rises to the top, where it collects as a floating layer. Other nitrides can be made in the same way. (943,132, Dec. 14, 1909.)

A New Type of Distillation Machinery.

BY WALTER E. LUMMUS.

A continuous distilling column of unique design recently built by the firm of Walter E. Lummus, of Boston, Mass., for the U. S. Forest Service, and known as a three-product distilling machine, has shown results on operation which appear to warrant a brief general survey of this system of distillation.

The acceptance of the machine referred to depended entirely upon the successful outcome of a series of trial runs with a variety of materials especially selected to demonstrate the amplitude of operation. The results achieved appeared creditable in every respect and seemed to meet with the requirements of the Forest Service as well as to justify the hopes of the builder. The machine is now set up at the new Forest Products Laboratory lately established at Madison, Wisconsin, and is being used in the refining of large samples of crude wood turpentine. This work is being carried on in connection with a series of experiments undertaken by the Bureau primarily intended to lead to a more thorough utilization of the waste material resulting from the lumbering of the Southern yellow pine.

Machine Versus Apparatus.—Practically all distilling apparatus has hitherto been designed to obtain profitable results through the skill and judgment of expert operators. The financial return from such operations varies therefore with the diligence and care with which the work is conducted. But in all the more progressive arts, automatic machinery is taking the place of hand work because more reliable and more economical. The object of the Lummus distilling machine design is a compact, self-contained, and automatic machine, capable of performing work equal to that of the most skillful operators and of performing the work.

Construction.—The accompanying cut illustrates a machine capable of turning out not less than four refined products, simultaneously and in marketable condition. It is a form better adapted to general manufacturing conditions than the one built for the Forest Service, which was specially designed for experimental purposes.

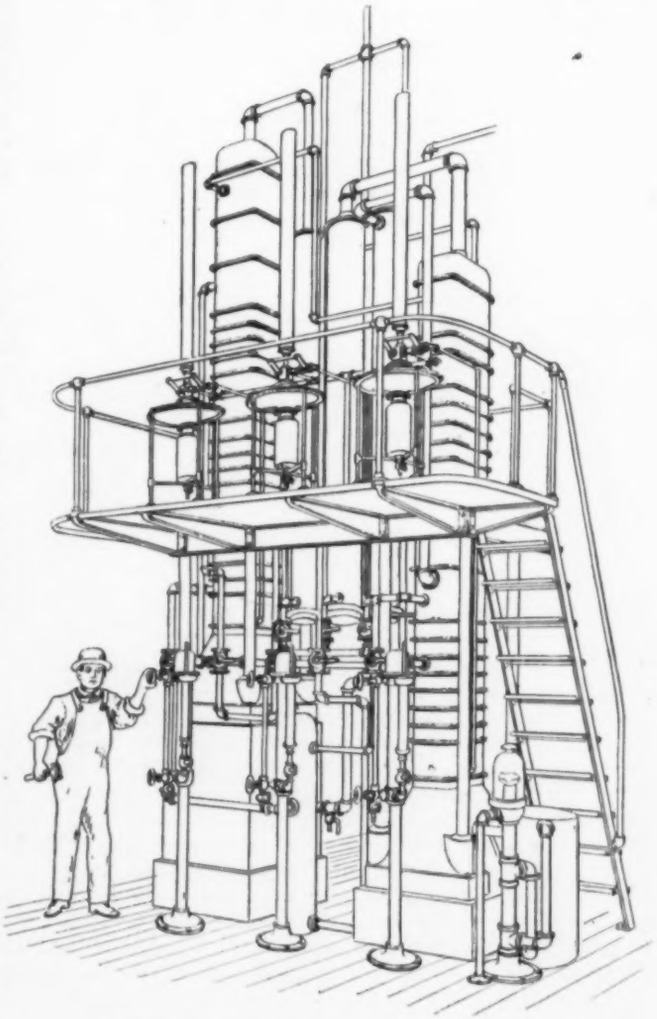
A complete description of this machine and the nature of the operations by which its work is accomplished would involve the study of multiple-effect cells and an understanding of the conflict of whirling masses of foam and vapor in what might seem hopeless complication. Really the process is one of orderly and selective evaporation and the apparatus has been reduced by scientific elimination to a sober and practical solution of what has been heretofore a vexatious and difficult task.

The machine as shown consists of two columns composed of unit sections made of suitable metals and built upon a unit system which provides for modifying the apparatus to the most favorable arrangement for the work to be performed. In the distilling chambers comprised in these columns are devices for causing the vapor to undergo fractional distillation and condensation, to the end that the ingredients of the crude material automatically assort themselves "like by like" and go their separate ways in the proportions in which they occur in the crude. Between the columns or upright series of chambers are suspended condensers at appropriate heights so that the products as they issue may flow by gravity into raised tanks from which, in turn, they may be drawn with convenience for shipping.

In front of the machine and at a height convenient for operation, are speed gauges or spiritometers especially designed for the purpose which show the rate of flow of each product, its gravity and temperature. Between the gauges is the switchboard or bank of operating valves, arranged with simple and methodical relation to the gauges so that the operator may observe his gauge and manipulate the regulation of the machine to the exact result desired.

The makers emphasize specially the simplicity of operation, the compactness, and workmanship of these machines in contrast with the complicated construction and enormous bulk of foreign apparatus intended to turn out similar products. In regard to compactness it may be well to add that the particular machine illustrated occupies a floor space about 3 ft. 6 in. x 7 ft. 6 in. and is 17 ft. over all. The capacity, on the other hand, is between 600 and 720 gals. of crude material in 24 hours, or 25-30 gals. per hour.

Operation.—Continuous distillation is not new in Europe, and there are simple continuous "beer stills" in modern American distilleries. The latter, however, are not depended upon to make finished products but only to concentrate material that is to undergo later more thorough rectification. The advantages of continuous operation will be obvious to everybody.



MULTIPLE-PRODUCT DISTILLATION MACHINE.

The distilling machine described is operated continuously and no boiler or kettle is employed, nor any system of tanks to hold liquors in various stages of completion. When once properly adjusted the material travels without pause through the various chambers, emerging from appropriate condensers according to composition. The operation may be continued indefinitely so long as a uniform supply of crude material, steam, and cooling water are maintained. Very sensitive regulators control the pressure and temperature and even the gradually diminishing head of the supply tank from which the raw material flows to the machine. To this feature of absolute control the uniformity and improved quality of products are largely due.

The most remarkable feature of this machine, however, and one which perhaps indicates an entirely new development of industrial engineering, is the feature of "multiple products."

Not only is the main product isolated in pure condition from the complex mixture in which it occurs as "crude," but advantage is taken of the favorable condition of the residue while yet hot to further assort the mixed gases in compartments designed for the purpose. In this manner each component of commercial value is likewise isolated and flows from its respective condenser in a manufactured condition.

Economy.—The economy of operation of these machines is especially worthy of attention. Since the raw material enters only as fast as the products leave, it follows that the quantity of material present in the machine at any one time can be reduced to that actually undergoing distillation and the steam and water consumption are correspondingly small. By a further economy, without compensating disadvantage of any kind, the raw material is heated to boiling temperature before entering the machine proper, with the waste heat of the products or effluents. In ordinary stills, a quantity of material sufficient for the entire operation must be maintained at the boiling point throughout the entire period, thus consuming many times the fuel required for evaporation, and no practical means has been found to utilize the heat of residues or distillates except in rare and unimportant instances. The machines, as built, have a capacity, according to size, far in excess of all previous records. The interaction of gases and liquids, upon which the efficiency depends is here produced by devices employing several new principles of distillation, and the superior yields and quality are partly due to these improvements.

Cost.—Not only is the cost of manufacturing with these machines far less than with the usual equipment of stills and tanks, but even the first cost of installation is less with a consequent saving of interest, depreciation and repairs. The Lummus distilling machines require no special building construction.

To sum up, there are two main distinctions between these distilling machines and the ordinary apparatus of well-known form:

Firstly, this new type may be termed a machine in every sense of the word, being compact, self-contained and automatic. It is especially designed to meet American conditions, which, owing to the high price of labor, tend directly toward automatic machinery.

Secondly, the elastic unit system employed in construction permits the adapting of these machines to the special requirements of any crude material. The most unusual demands may be met with an exactness that has heretofore been considered impossible.

These two points alone explain the ready application of the Lummus machines to a vast number of distillation problems. Wood alcohol, turpentine, petroleum, fusel oil, tar oils, etc., which for years have been manufactured by tedious periodic processes are with these machines easily separated in one continuous operation into their ultimate constituents or groups of constituents, conforming to commercial specifications. Furthermore, products that have heretofore defied economical manufacture on account of prohibitive cost of periodic refining, are now brought within the range of profitable production.

Slime Filtration.

A new filter press placed on the market by the Sweetland Filter Press Company (Ernest J. Sweetland, General Manager, Union Trust Building, Los Angeles, Cal.) is of special interest, as it is a pressure filter, but is designed to have the chief advantages of vacuum filters, especially from the point of accessibility and the avoidance of having to open the filter chamber for each discharge.

The general appearance of the Sweetland filter press is shown in Fig. 1. The press is assembled on a slant of about one inch per foot, while the filter leaves hang vertically.

In order to gain the advantages of accessibility, the Sweetland filter press (Fig. 1) is built up of a plurality of sections. Each section is practically complete in itself, and, all being alike, they are interchangeable. In this way the capacity of the filter may easily be enlarged at any time it should be desired, for the enlargement of the filter merely requires the addition of more sections, and any number may be inserted by lengthening the tie rods sufficiently to accommodate them.

Each section consists of a cast-iron frame, the upper portion of which is in the form of a rectangle, while the lower walls

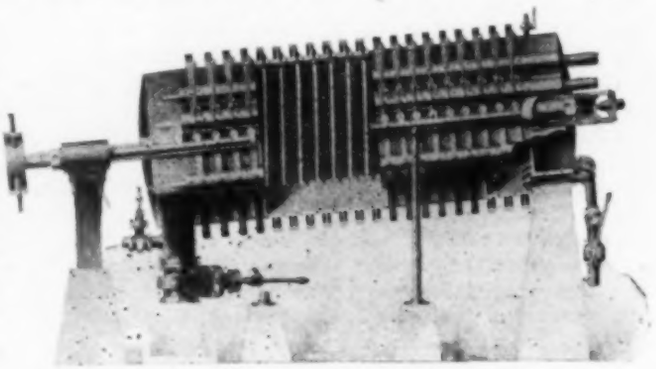


FIG. 1.—FILTER PRESS.

converge, forming a triangular opening in the lower part of the frame below the filter leaf (Figs. 2 and 3).

A special feature is the form in which the frames are cast; they have an I-beam section (instead of a rectangular or square section, as the old type of frames). In other words, there is a strengthening rib on the outer surface of the frame. This rib is in the form of a thin web of metal cast upon the outer wall of the frame, at right angles to it, with a bulb cast upon its outer edges. This construction results in great strength and light weight. While the main wall of the furnace is only about ¾ in. thick, the frames stand an internal pressure of 100 lb. per sq. in. without injury.

Fig. 2 shows one of the frames. Both sides of the frames are planed to a smooth, flat surface. On one side a groove ¼ in. deep and ⅜ in. wide is cut into the frame, into which a rubber gasket ⅜ in. square is inserted, which abuts against the flat side of the adjacent frame. This gasket is firmly impressed within

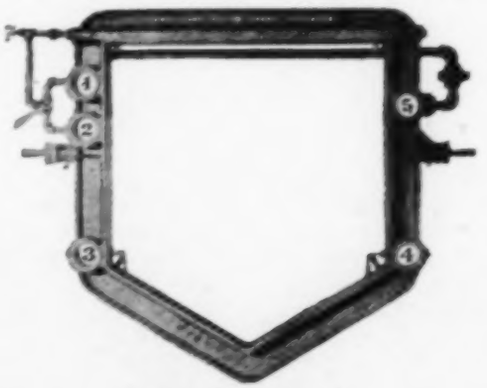


FIG. 2.—FRAME SECTION.

the groove, and forms a water-tight joint up to 100 pounds pressure per square inch.

The spray pipe is clearly shown in this figure, extending horizontally across the top of the frame just below the upper wall.

There are five circular openings or passageways through each frame. When the frames are aligned in position in the press these passageways form continuous conduits throughout the length of the press and terminate in pipe connections at the upper head.

When the filter leaf is in position (Fig. 3) it is connected by a union to the frame at the upper right-hand corner. The upper

left-hand corner is held in place by a small clip, while a pair of lugs attached to the frame support the two lower corners. The union and pipe connection at the upper right-hand corner brings the interior of the filter leaf into communication with the frame. There is a horizontal passageway through the top of the frame, which extends from one side to the other and terminates at the left-hand side in a pipe connection. Other pipe connections are provided leading into conduits Nos. 1 and 2 (Fig. 2), and join in a lever-handled three-way valve, the third opening of which connects with the pipe communicating with the filter leaf. The filter leaf may therefore be brought into communication with either conduit No. 1 or No. 2, according to the way the three-way valve is turned. The arrangement is plainly shown in Fig. 2.

A test-cock is shown on the pipe connection, the object of which is to provide separate communication with each leaf, so that in case a leak should occur at any time the filter leaf containing it may be immediately located and shut off from further filtration until such time as it is convenient to repair the leak or replace the leaf.

During filtration the three-way valve is turned to allow the filtrate to drain into conduit No. 2. At all times conduit No. 1 is kept filled with compressed air under moderate pressure, or water under a 12 to 2-foot head supplied by a pipe connected with the upper head of the press, and when it is desired to discharge the slime cake from the filter leaf the three-way valve is turned to allow the compressed air or water to enter the interior of the filter leaf from conduit No. 1, which causes the cake to drop instantly.

Each frame is provided with two small nozzles located near the lower corner of the leaf, which communicate with conduits Nos. 3 and 4 (Fig. 2), respectively. The object of these nozzles is to remove the last traces of slime which may adhere in the corners of the press after discharging. If the pressure of the pump is turned into these conduits for about 30 seconds after the bulk of slime has been discharged, the body of the press is left absolutely clean for the following cycle of operations.

Conduit No. 5 (Fig. 2) supplies the spray pipes with liquid. Each spray pipe is provided with an individual valve connection between the pipe and conduit, so the full pressure of the water supply available may be turned into all the spray pipes simultaneously, or into a few of them at a time, as may be desired. These spray pipes are not used in the ordinary cycle of operations, but for occasionally washing the filter leaves.

The filter leaf which is shown in place in the frame (Fig. 3)

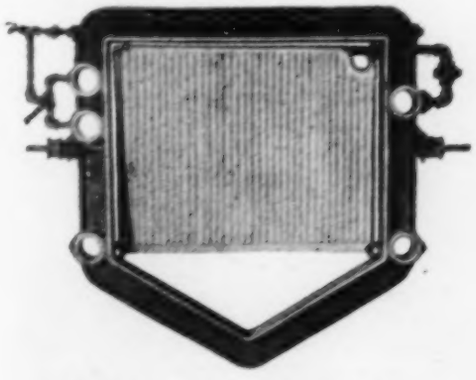


FIG. 3.—FRAME, WITH FILTER CLOTH.

comprises a series of vertical corrugated wooden strips covered with a layer of canvas, which forms the filtering medium. Each leaf is provided with a drain pipe terminating in a union outside of the leaf. To place the leaf in the frame it is only necessary to connect the union after inserting the leaf between the lugs at the other three corners. The filter leaves (standard size, 3 ft. 2 in. by 4 ft.) are so easily taken out and replaced in the frames that two men can easily remove a set from a 40-frame press, replace them with fresh ones, and have the press ready for operation again in about an hour.

The spray pipes are clearly shown in Fig. 1 between the filter leaves in the upper part of the press, and the discharge nozzles are shown entering from the upper head of the press, in line with the lower walls of the frame. There are five discharge nozzles, each ½ in. to ¾ in. in diameter, depending on the length of the press. All are controlled by a single quick-open-

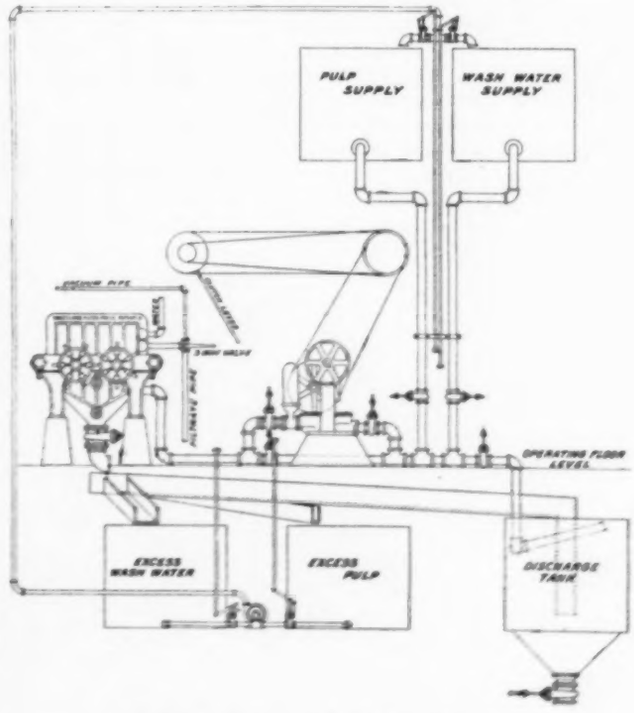


FIG. 4.—ARRANGEMENT OF FILTER PLANT.

ing valve on the discharge line of the plunger pump. The purpose of these nozzles is to flush the slime residues from the press when caused to fall from the filter leaves.

The discharge valve is plainly shown in this figure, attached to the hopper on the lower head of the press. This is an 8-in. quick-opening valve, and is used whenever it is desired to empty the press either of excess pulp or wash water, or when discharging the residues.

When the press is installed, there are three launders below the discharge valve; one for excess pulp, one for excess wash water and one for discharge. By means of a revolvable switch (not shown in Fig. 1) the discharge from this valve may be directed into any one of the three launders.

A suitable arrangement of a filter press plant intended for 100 tons' daily capacity is shown in Fig. 4. In this arrangement the filter press and pump which operates it are placed on a level below the pulp-supply tank and wash-water supply tank, and on another level just below the filter press are the excess pulp tank and excess wash-water tank, and a small centrifugal pump to return the excess pulp and wash water to the supply tanks on the upper level. By this arrangement the filter press is both filled and emptied by gravity, and the necessity of a large pump for rapidly filling and emptying the press is eliminated. A difference in level of 25 ft. between the supply tank floor and the excess tank floor is sufficient to permit of this arrangement, although more fall can be used to advantage if available.

The pump shown on the operating floor level is a triplex outside-packed pump, the size of which varies according to the size of the installation. This pump serves two purposes, viz., that of forcing the sludge or wash water into the press during filtration and washing, and to deliver water to the discharge nozzle while the press is being discharged. As a greater volume is required

during discharging than during filtration, the pump is so arranged as to have two speeds.

The filter press is first filled with pulp from the pulp supply tank through the by-pass under the pump.

As soon as the press is filled the low-speed clutch of the pump is thrown in, which forces the pulp into the press under 40 or 60 lb. pressure. A relief valve is provided, which keeps the pressure from rising above the desired point. As the pulp is forced in, the clear filtrate passes through the canvas of the filter leaves and out through the conduit No. 2 into the filtrate pipe, and the slime is deposited on the surface of the filter leaves, forming a cake from 1 in. to 2 in. in thickness, depending upon the length of time the operation is continued.

When the cake has reached the desired thickness the pump is stopped and the three-way valve on the filtrate pipe turned so as to bring the vacuum pipe in communication with the interior of the filter leaves, the object of which is to hold the cakes intact on the filter leaves while they are relieved of pressure. A vacuum of 5 in. only is rquired to hold the cakes in place. The swivel joint or switch, just below the discharge valve, is now turned to direct the flow into the launder leading to the excess pulp tank, and the discharge valve is opened to allow the excess pulp to rapidly drain out the press.

The discharge valve is then closed and the press filled with wash water (or barren solution, as the case may be) through the by-pass. When the chamber of the press has been filled the pump is again started at low speed, pumping the wash water through the cakes to displace the dissolved gold or silver values. The displacement is very rapid under 40 to 60 lb. pressure, and usually takes about ten minutes.

The excess wash water is drained from the press into the excess wash-water tank, and the slime cakes may now be discharged as follows:

The switch below the discharge valve is turned into the discharge launder. The valve on the suction line of the pump, which leads to the cone-bottom discharge tank, is opened. The pump is started at full speed, which delivers a stream of water from the top of the discharge tank to the discharge nozzles under about eighty pounds pressure. There are five of these nozzles, all of which are in operation at the same time, and a stream of water of high velocity is caused to sweep the bottom of the press. With these streams in operation all that is necessary is to turn down the lever handle upon each of the filter press frames, which causes the water or compressed air in conduit No. 1 to enter the filter leaves, causing the cakes to fall from the leaves to the bottom of the press, one at a time. As fast as they fall they are partially disintegrated and carried away by the streams from the nozzles into the discharge launder, which deposits the residues in the cone-shaped tank.

Each cake is given about ten seconds to fall before the next one drops, which is ample time for the leaf to become thoroughly cleaned by the reverse current through the pores of the canvas, and for the cakes to be carried away. Fresh water is not required for use in discharging the residue, but the discharging water is merely circulated from the top of the cone-bottom discharge tank through the bottom of the filter press back to the discharge tank again.

In time a cleaning of the canvas becomes necessary, due to the formation of a deposit of lime and silica, etc., which requires acid treatment and scrubbing of the canvas. For this purpose the Sweetland Filter Press Company have developed a special revolving brush for cleaning the filter leaves.

From tests made with the Sweetland filter brush it appears that the rate of filtration is practically in direct proportion to the pressure at which the liquid is applied to the filter surface.

Brazing Plant.

By Richard N. Hart.

Unless a broken iron casting is replaced by a new piece, there is one kind of repair that will restore its original strength. This is by brazing. A well-brazed fracture in cast iron adds from 10 to 25 per cent to the initial strength. This is because the brass amalgamates or alloys with both broken surfaces. Brass is stronger and tougher than iron.

It is generally much cheaper and quicker to braze a fractured casting than it is to buy a duplicate part. For this reason some of the larger foundries and machine shops have brazing apparatus of their own, or an independent brazing shop will be set

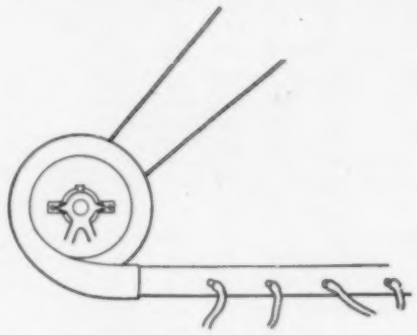


FIG. 1.—BLOWER AND AIR TUBE FOR BRAZING PLANT.

up in the midst of a community of factories. Such a shop can make a large margin on its repair works and save a still larger sum for the manufacturer. Occasionally, when the part to be repaired is part of an apparatus of foreign make, the saving in time, anxiety and money may be very great. Every mining camp of any importance, at a distance from supplies, should certainly have a brazing outfit.

The brazing outfit here described can be installed in large shops, or can be set up in an independent repair shop. It consists of torches using gas and air blast, a makeshift brick oven and miscellaneous tools, such as a vise, hammer, files, wire brushes, etc.

The city gas at a pressure of about two pounds is used. The air blast is furnished by a rotary fan playing into a long cylinder, as shown in Fig. 1. The fan is driven by a 2-hp or 3-hp motor, and gives a pressure of six to eight ounces. As many as six burners can be attached to the air cylinder at once; for heating large castings this may be necessary.

The torch is a two-way gas pipe, Fig. 2, mounted on a tripod with a swivel joint and connected to the gas meter and the air cylinder by two wire-wound rubber tubes. These tubes should

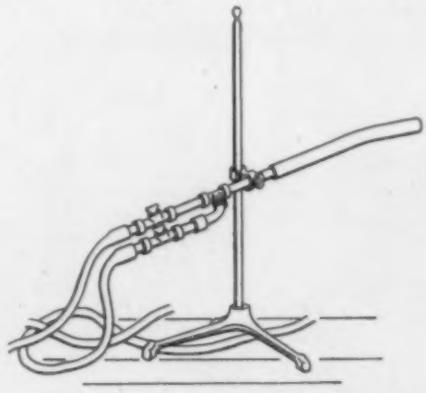


FIG. 2.—AIR-GAS TORCH FOR BRAZING.

be long enough to allow the burners to be moved about freely.

The furnace may be made of loose brick built up around the casting so as to hold it in position and to contain the heat well. The table can be protected by asbestos cloth. The roof of the brick furnace can be held up with iron bars, as shown in Fig. 3, unless the casting is small enough to allow the roof to be made of one brick. The front of the furnace is left open and the torches are played directly into the front. No vent is needed

for the gases of combustion, because the crevices between the bricks are large enough, and the heat is thus better retained.

Briefly stated, a broken casting is repaired as follows: The fractured surfaces are brushed with a stiff wire brush, and the surfaces near the fracture are cleaned of paint, rust and grease with file, brush or waste. Paint the fracture surfaces with an emulsion of bronze filings, sal ammoniac, borax and water. Fit the broken pieces together and build the bricks up around the

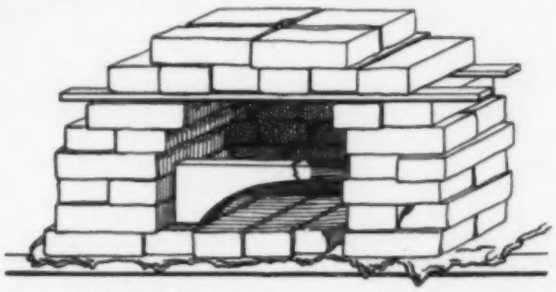


FIG. 3.—MAKESHIFT BRICK FURNACE FOR BRAZING, SHOWING BROKEN CASTING IN POSITION.

casting. Be sure that the pieces are fitted as firmly and closely as possible. They may easily separate slightly from the motion of the bricks while you are making the furnace. Next bring the mouth of the burner up to the open front of the furnace, point it toward the break in the casting and light the gas.

Regulate the air-gas mixture so that the flame has a distinct blue cone—*not purple*. If the casting is large, use two, three or more torches. When the metal is red-hot, apply additional flux to the fracture joint with an iron rod and rub it well along the crack. The flux is calcined borax or a mixture of calcined borax with sal ammoniac, salt, etc. Then shovel on a spoonful of brazing spelter, which will melt and run wherever the flux has cleaned the metal. Use plenty of spelter. Half-and-half brass may be used, or, better still, a brass of 70 per cent copper, which will be stronger and less apt to lose its zinc by burning.

The casting may now be cooled and cleaned. The cooling, like the heating, must be very carefully done if the casting is

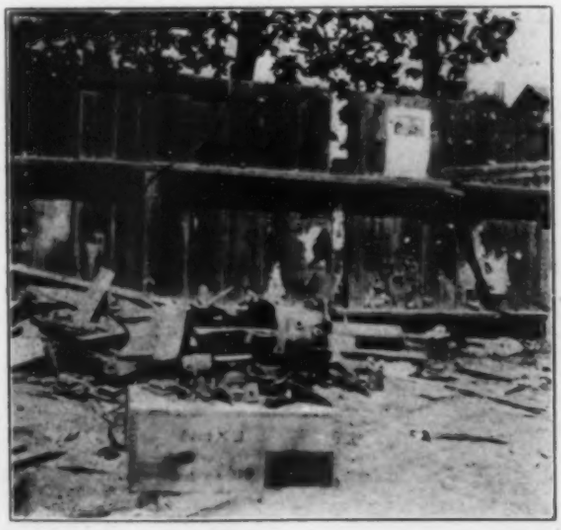


FIG. 4.—BROKEN MOTOR CYLINDER.

large or delicate or liable to break through expansion. Cast iron will gain in strength from 10 to 25 per cent by brazing, and will not break again in the same place. Steel, malleable iron, brass and bronze can also be brazed.

Figs. 4 and 5 show a motor cylinder that was broken into 33 pieces and brazed together by the Cast Iron Brazing Company, of Los Angeles, Cal. It was necessary to hold it together with wire. The entire operation took a little over two days. Care is necessary in heating up motor cylinders, otherwise they will have to be reground because of unequal expansion. Or if the flame is oxidizing (purple) it will rust the bore of the cylinder.

The cost of the apparatus can be brought within $400, or even $300. If the outfit is put up in the machine shop, and the power is taken direct from the shafting, a $150 motor can be dispensed with. If the outfit is put up at a mine, the gasoline blower from the assay office will take the place of motor, fan, cylinder and torch. In fact, it is a question whether a good gasoline torch would not be as safe, cheap and easy going as the outfit already named.

The Cast Iron Brazing Company estimate their gas and power cost at from 15 to 20 cents per hour for one burner, and about 10 cents additional for each extra burner. A casting of, say, 25-pound weight can be brazed in about one hour, counting cleaning before and after. The labor cost would be about 40 cents. The approximate cost of a one-hour repair job, provided the plant is in steady operation nine hours a day, is roughly as follows:

6 per cent interest on $400 apparatus	$ .01
10 per cent yearly depreciation on same	.02
Rent $200, taxes, insurance	.10
Power and gas	.20
Labor	.40
	.73

Between this cost and the average value of the casting to be

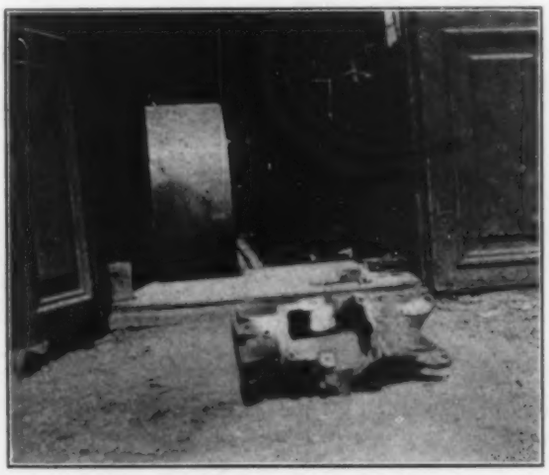


FIG. 5.—REPAIRED MOTOR CYLINDER.

mended there is a very wide margin, allowing a substantial profit.

Whenever this brazing outfit is needed it is badly needed. A mine, or any isolated plant using machinery, is sure to be tied up occasionally by the breaking of parts. Delay is always tremendously costly and exasperating. A trifling outlay, such as just described, will prevent breakages from demoralizing the plant.

Gas Velocity Recording Device.

A new velocity gauge for gases, which is being placed on the market by the Hydro Manufacturing Company, 720 Lewis Block, Pittsburgh, Pa., is interesting, as it is based on a novel principle.

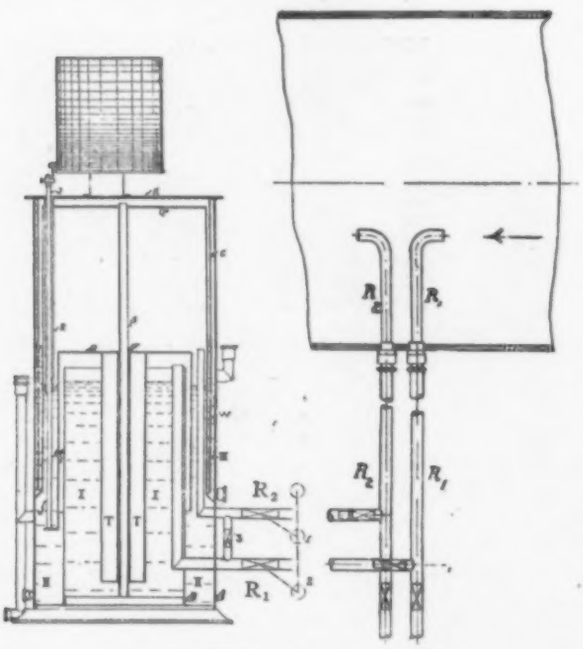
The gauge is shown in the adjoining diagram. The gas, the velocity of which is to be measured flows through the flue shown on the right hand, in the direction of the arrow. Within the path of the gas the ends of two pipes, R_1 and R_2, are placed. The open ends of both pipes are bent, but in opposite directions. That of R_1 is bent opposite to the flow of the gas, that of R_2 is bent in the same direction as the flow of the gas. This causes a difference of dynamic pressure in the two pipes R_1 and R_2,

and this difference of pressure depends solely on the velocity of the gas. By measuring the pressure difference, we therefore get an indication of the velocity of the gas.

The pressure difference is measured in the apparatus shown at the left. It consists in the main of two cylindrical compartments, one within the other. Both are partly filled with liquids, of which the inner one serves as operating medium and the outer one as a water seal. The inner liquid *I* carries a float *T* to the top of which is fastened an inverted cup-shaped cylinder *D*, which dips with its lower open rim *M* into the outer sealing liquid *II*. In this manner two independent gas chambers are formed, both of varying capacity, depending upon the depth of the immersion of the bell, which in turn depends directly upon the variation of pressure within and without.

A rod *S* that is centrally fastened to the bottom of the main cylinder serves as a guide for the operating bell. Its upper end passes through the removable cover plate *P* of the apparatus.

The movement of the float or bell is carried outwardly by means of a second rod *J*. This rod is hinged to the lower portion of the bell and it passes through a tube *R*. The top end of this tube is secured air-tight to the cover, and its lower end dips into the sealing liquid *II*. The rod carries at its upper end a writing pen which bears against a rotating cylinder operated by clockwork, and this serves to trace the varying differential pressures of the system upon a chart.



GAS VELOCITY METER AND RECORDER.

An additional water seal *W* is provided for the cover by a cylindrical portion *C* extending therefrom downwardly into the sealing liquid *III* of an annular space, formed within the upper portion of the main body.

Communication between the gas main and the apparatus is established by two pipes R_1 leading into the air space below and R_2 leading into the space above the movable bell. Each of these pipes is provided with a stop-cock, 3, and both cocks are linked together to make them work simultaneously. By these means it is made possible to distribute the pressure to both sides of the bell at the same time. A three-way cock serves to make communication of both air spaces in the apparatus with the atmosphere.

The filling of the three independent liquid chambers is accomplished through pipes which lead out from the main body of the apparatus. Plugs are provided for at the lower end of these pipes, which allow easily draining the receptacle if this should be necessary.

The velocity of the gas in the flue causes, as explained above, a difference of pressures in the two pipes R_1 and R_2, and this pressure difference causes the movement of the bell which is transmitted to the writing pen on the recording device.

This recording velocity gauge should be useful in gas works, producer gas installations, coke oven plants, blast furnace plants, and in general in metallurgical and chemical works in which the success of the process depends on a definite rate of supply of air or gas. Naturally the instrument is also useful in boiler plants, where it is employed to check the correct air supply for the most economical combustion of the coal; in this case the record chart indicates every operation of the fireman or of the automatic stoker, and serves as a guide to the fireman in which manner to obtain the best results.

Notes.

The **Zaremba Company,** builders of evaporators and chemical machinery, announce the removal of their engineering department and Eastern office from Philadelphia, Pa., to 1044 Ellicott Square, Buffalo, N. Y.

Corubin.—A valuable by-product, obtained in the production of chromium by the thermit process, is known as "corubin" (registered trade-mark), which has proved to be one of the best abrasive materials obtainable. It is used extensively abroad in the manufacture of high-grade emery wheels, emery cloth, and for other purposes where a first-class abrasive is required. Arrangements have now been completed for supplying the American market with this material, and the Goldschmidt Thermit Company, 90 West Street, New York, will be glad to give further particulars.

Helion Resistors.—The Parker-Clark Electric Company announces the retirement of Mr. W. G. Clark from his connection with the Company and the election of Dr. George M. Miller to the office of president and director. Prof. H. C. Parker continues at the head of the laboratory work. The offices of the company have been removed to the Lincoln Square Building, 1960 Broadway. The company is now placing on the market the helion resistors which are being made by the action of silicon on carbon for use as resistors in electric heating apparatus, etc. Anticipating a large demand for these resistors, the company plans to start a factory for their manufacture as soon as a good location can be selected.

A Study of the Open Hearth is the title of a very interesting little book just issued by the Harbison-Walker Refractories Company, of Pittsburgh. It is a concise and very clear treatise of the open-hearth furnace and the manufacture of open-hearth steel. The first and principal chapter deals with the open-hearth furnace and gives many useful data on refractories used in it. The other four chapters deal with fuels, the acid open-hearth process, recarburization, the basic open-hearth process and special processes (Talbot, Monell, Bertrand-Thiel, Duplex). Some parts of the book deserve a more detailed review, which we reserve for a future issue.

The Palmetto Phosphate Company, of Tiger Bay, Florida, has recently put in operation its new producer-gas electrical generating plant. The first of the three units was started Nov. 23, the second Nov. 26 and the third Dec. 2. Briefly, the equipment consists of one 1100-hp gas-producer plant in three units, three 18 x 24-in. single-tandem gas engines direct connected to 200-kw, 2300-volt generators together with the necessary exciters, transformers, switchboards, etc. The entire plant is now electrically operated. The Allis-Chalmers Company are the builders of the apparatus.

Oxy-Hydrogen Welding and Cutting.—While the oxy-acetylene process has been introduced to a large extent in this country, both for welding and cutting, the older analogous process in which the oxy-hydrogen flame is used, and which is employed largely in Europe, has been comparatively unknown in this country. A change in this condition is now to be made by the American Oxhydric Company, of Milwaukee, Wis., the owners of the American patents of the Société l'Oxhydrique

Internationale for apparatus for the welding and cutting of metals by means of the oxygen-hydrogen flame. The American company thus has the benefit of years of experience with this apparatus in France, Germany, Belgium, Italy and Russia. The company manufactures the oxygen and hydrogen gases by electrolysis of water.

The **West Steel Casting Company,** Cleveland, O., who for the past three years have built up quite a large trade in medium and small steel castings made by the converter process, have lately enlarged their plant and are now installing crucible furnaces which will supply special small castings demanded by the automobile and general trade. The converter metal will be utilized as heretofore for casting of West's "electrical steel," which is largely used for dynamo frames and other electric castings. This same metal is also found valuable on account of its good strength and ductility for blank gears and general steel castings. The officers of the company are Mr. Ralph H. West (son of Mr. Thos. D. West), president; Mr. Everett W. Pike, vice-president, and Mr. David P. Lansdowne, secretary.

Cyanide Process in Mexico.—The first plant to be built in the famous Parral gold mining district of Mexico for the treatment of ore by the cyanide process has been decided upon by the Palmilla Milling Company, of Parral, State of Chihuahua. The plant is designed primarily for treating the ore of the Palmilla Mine; but will also be arranged for treating custom ore. The "Vetacolorada" or "red-vein" of the district is capable of producing many thousands of tons of ore of reasonable grade which can be treated successfully by cyaniding and concentration. The cyanide part of this new plant will include the best in filter and air agitation tank construction, though these latter details have not yet been decided upon. The machinery has been purchased from Allis-Chalmers Company, Milwaukee. Stamps are of latest design. The ordinary accessories such as grizzlies, crushers and conveyors are provided. All wearing parts were specified to be of specially tough steel. Three tube mills are included, which will be 5 ft. in diameter and 22 ft. long and will have the latest features of construction, such as automatic spiral feeders and ribbed liners.

Mr. **Elmer A. Sperry,** as has been well known, did a great deal of work several years ago in collaboration with Mr. C. P. Townsend to reduce to practice what is now well known as the Townsend cell for the electrolytic production of caustic soda and chlorine. This cell has now been in continuous successful use on a large scale for several years at Niagara Falls. It has not, however, been generally known until the present announcement that during this time of collaboration of Mr. Sperry with Mr. Townsend several further processes were developed and tested on an extensive scale. These include processes for the electrolytic treatment of ores in a fused bath, processes for the production of lead oxides, white lead (of low or high oil absorption), and pigments; also a process for tin ore treatment. These processes are said to have all the same characteristics as the caustic soda and bleach process of Townsend, namely, low first cost, directness of application, and economy of maintenance and power consumption. Further details concerning these processes may be had from Mr. E. A. Sperry, 40 Wall Street, New York City.

Obituary.

John Wellington Nesmith, president of the Colorado Iron Works Company, aged 76, died on Friday morning, December 17, of heart failure following an operation for gall stones. He had been in splendid health and attending to his usual business occupations until Monday, and his death came as a shock to his relatives and friends. Mr. Nesmith had been identified with the progress of the West from the early days, coming to Colorado in 1860, engaging in mining and milling in Gilpin County. He subsequently became interested in Langford & Company's foundry, the nucleus of the Colorado Iron Works, of which he became manager in 1879 and president in 1886. In the advancement in ore-treatment methods he took an active interest, and specialized in the blast-furnace smelting of lead and copper ores from the time of the first application of the process, achieving a world-wide reputation for the Colorado Iron Works in this line. Mr. Nesmith was a past master Mason and a past high priest of the Royal Arch Masons, vice-president for Colorado of the National Association of Manufacturers, a member of the American Institute of Mining Engineers, the Colorado Scientific Society, and of various other organizations.

Digest of U. S. Patents.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ELECTRIC FURNACES (Continued).

No. 643,898, Feb. 20, 1900, Harry M. Hill, St. Louis, Mo.

Resistance type. Oven for dental use, assaying and melting; an inclined crucible of fire clay or magnesia, having an external spiral groove receiving a doubled platinum wire of 36 gage, the groove being deeper than the diameter of the wire. The crucible is surrounded by windings of asbestos or mineral-wool paper, applied moist and baked. Before applying this paper the crucible is heated, and shrinks. The wire resistor is then rewound. The crucible is closed by a plate of refractory material, also grooved externally and receiving a resistance wire.

No. 645,284, March 13, 1900, Edward G. Acheson, Buffalo, N. Y.

Resistance type, for heating, roasting, and especially for purifying and graphitizing carbon. The furnace chamber is rectangular, two sides sloping at an angle corresponding to the angle of repose of the charge-mixture to an opening at the bottom, beneath which is a screw conveyor rotating in a water-cooled tube. Opposed electrodes pass through the walls of the chamber near its top. For the production of 500 lb. of graphite from coke per hour, the electrodes have a cross-section of 1250 sq. in. and are spaced apart 20 in., passing a current of 37,500 amp. They are composed of a number of rectangular bars arranged in contact. Pipes carrying water for cooling the product pass transversely through the lower portion of the chamber. The furnace has a suitable supply hopper, and vents which may lead to condensing chambers.

No. 647,614, April 17, 1900, Marcus Ruthenburg, Philadelphia, Pa.

Furnace for smelting and refining ores, comprising a chamber in the form of an inverted quadrilateral pyramid, formed of slabs of refractory material, opening beneath into a separate hemispherical crucible. Horizontal carbon electrodes pass through water jackets into the opposite sides of the chamber, preferably carrying 1000 amp per square inch of cross-section. The molten product, gravitating into the crucible, overflows through a spout at one side into a wheeled receptacle.

No. 650,040, May 22, 1900, Ernest W. Engels, Essen, Germany.

Bricks, slabs and plates of silicious, clayey or other refractory materials are rendered fire and acid proof by coating them with a layer of carborundum or other carbid, melted or fused on by an electric arc, for example, one of 200 amp at 110 volts. Either the pre-formed carborundum or the materials for producing it, sand and charcoal, may be placed on the surfaces of the bricks and then fused, uniting with the brick.

NEW BOOKS.

FINLAY, J. RALPH. The Cost of Mining. An exhibit of the results of important mines throughout the world. 424 pages, illustrated, maps. Price, $5.00 net. New York: McGraw-Hill Book Company.

BASKERVILLE, C. General Inorganic Chemistry. 364 pages. Price, $1.50. Boston: Heath.

BASKERVILLE, C., and CURTIS, ROBT. W. Laboratory Exercises to Be Used in Conjunction with General Inorganic Chemistry. 57 pages. Price, 40 cents. Boston: Heath.

KAHLENBERG, LOUIS. Laboratory Exercises in General Chem-

istry. Second edition, revised. 208 pages. Price, 90 cents. Madison, Wis. (L. Kahlenberg, University of Wisconsin.)

PECKHAM, STEPHEN FARNUM. Solid Bitumens. Their physical and chemical properties and chemical analysis; together with a treatise on the chemical technology of bituminous pavements. 332 pages. Price, $5.00. New York: M. C. Clark Publishing Company.

COYLE, D., and HOWE, F. J. O. Electric Cables; Their Construction and Cost. 473 pages, illustrated. Price, $5.00 net. New York: Spon & Chamberlain.

FRANKLIN, W. SUDDARDS. Electric Waves. An advanced treatise on alternating-current theory. 324 pages. Price, $3.00 net. New York: Macmillan.

MCCLUNG, R. K. Conduction of Electricity Through Gases and Radioactivity. A textbook with experiments, with 78 illustrations. 261 pages. Price, $1.50 net. Philadelphia: P. Blakiston's Sons & Company.

REEVE, SIDNEY ARMOR. Energy, Work, Heat and Transformations. 238 pages. Price, $2.00 net. New York: McGraw-Hill Book Company.

MALCOLM, C. WESLEY. A textbook on graphic statics. 328 pages. Price, $3.00. New York: M. C. Clark Publishing Company.

MOLESWORTH, SIR GUILFORD LINDSEY. Metric Tables. Fourth edition, revised and enlarged. 104 pages. Price, 80 cents. New York: Spon & Chamberlain.

RUSSELL, G. E. Textbook on hydraulics. 190 pages. Price, $2.50. New York: Henry Holt & Company.

STEPHENS, J. RITTENHOUSE. Principles of Accounting. 91 pages. Price, $1.25. Kansas City, Mo.: John R. Stephens, Keith & Perry Building.

HATHAWAY, C. E., and GRIFFITH, JA. BRAY. General expense and cost summaries; instruction paper. 70 pages. Price, 50 cents. Chicago: American School of Correspondence.

BOOK REVIEWS.

A Course in Inorganic Chemistry for Colleges. By **Lyman C. Newell,** Ph.D., Professor of Chemistry in Boston University. Half-leather bound, 604 pages, 89 cuts. Price, $2. Boston: D. C. Heath & Company.

General Inorganic Chemistry. By **Charles Baskerville,** Ph.D., Professor of Chemistry in the College of the City of New York. Cloth, 357 pages. Price, $1.50. Boston: D. C. Heath & Company.

A Laboratory Manual. By **Dr. Charles Baskerville** and **Dr. Robert W. Curtis.** Bound, 64 pages. Price, 40 cents. Boston: D. C. Heath & Company.

The absolutely best chemical text-book will never be written, so there is always room for a better one.

The books noted above are faithful attempts to improve on existing text-books, in the teaching of first-year college students. Of the first, we may say that it is a very carefully written work, following the style of the best of the older text-books, is a mine of information and is full of clear and convincing explanations; while not breaking away from the methods of the older text-books, we may say that it is certainly a most excellent *vade mecum* for the earnest, industrious student.

The second book is a "departure," an attempt to part company with the older type of text-book. The prevailing idea is "to accomplish the most in the least time." To this end, statements are short and precise, the style is even "sketchy," but to a bright student it should be attractive and inspiring. We would rather have seen the motive of the work "the accomplishing of the best use of the time available." The other principle is likely to result in superficiality, even in "cramming." But Dr. Baskerville's book and methods, under the charm of the personal enthusiasm of a similar teacher, will probably result in fascinating many young men with chemistry, particularly students of the quick, bright, city type, with which many schools and colleges have principally to deal.

Dr. Newell's book is conservative, excellent, reliable, and highly instructive; Dr. Baskerville's is radical, at places brilliant, not so reliable, but inspiring. The teacher will at once perceive that each book will be the best for certain classes of students.

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The Elements of Metallography. By **Dr. Rudolf Ruer,** privat-docent at the University of Göttingen. Authorized translation by C. H. Mathewson, Ph.D., Instructor in Metallography at Yale University. Octavo, xiv + 342 pages, 119 illustrations. Price, $3 net. New York: John Wiley & Sons.

One is surprised and pleased to find that this work is not another contribution to the merely microscopic study of metals and alloys; in fact, the microscope is kept in the background throughout, being used for "supplementing the results of thermal investigation." The basis of the treatment is the thermal study of melting points and cooling curves, with all the deductions therefrom.

The physico-chemical theory is carefully and clearly explained. The one-component, two-component and three-component systems are systematically considered, the discussions being very lucid and interesting. This forms the bulk of the book.

Another chapter on "Practice" gives the technique of the thermal measurements, some very good advice on pyrometry and a brief description of preparing microscopic sections.

Altogether, it is a book for the thorough study and mastery of the best methods of investigation of metals and alloys. We agree with it entirely in putting thermal investigations in the first place and the microscope in the second place in the scientific study of most metals and alloys. We wish the work the acceptance and success which it richly deserves.

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Lo Zinco. By Prof. **Roberto Musu-Boy.** 16mo., 230 pages, 10 illustrations, 4 diagrams. Price, 3.50 lire. (Retail price in New York, $1.25.) Milan: Ulrico Hoepli.

A well written, neatly printed, cheap book, which will undoubtedly advance the knowledge of the metallurgy of zinc in Italy. Parts I and II deal with the properties and metallurgical extraction of zinc, respectively; part III with the zinc mines of different countries; part IV with the uses of zinc and its alloys; part V with economic conditions in the zinc industry, and an appendix with the manufacture of zinc white.

While there are many shortcomings in the presentation, as far as concerns practice in other countries, there is much to be learned from the work, particularly concerning the zinc industry in Italy. We find, for instance, that Italy stands second to Germany among European countries in the value of zinc ore production, but behind six others in value of metallic zinc produced. This is because of the lack of cheap fuel and points the way to a large zinc industry as soon as the electric zinc furnace is through its "Kinderkrankheiten." About all that is said on this subject is that the furnaces of Casoretti and Bertani, working some years ago near Bergano, have a production of 3.25 tons of zinc per hp-year, using 250 kg of carbon for reduction per ton of ore and extracting 90 per cent to 92 per cent of the zinc. The author should have given considerably more space to the proposed electric zinc furnaces, because of their great prospective importance to his country.

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The Wellcome Photographic Exposure Record and Diary, 1910. In cloth, 272 pages. New York: Burroughs, Wellcome & Company.

The 1910 edition of this photographic almanac contains a large amount of useful information for amateur photographers with respect to details of methods of development and exposure, the Wellcome exposure calculator chart being included. The almanac is exceedingly handsome.